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Engineering and Mining Journal

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NEW YORK LONDON

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June, 1932

McGraw-Hill Publication

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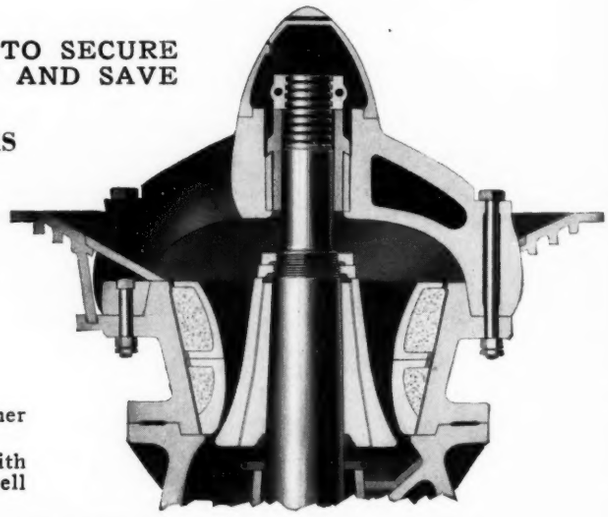

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June, 1932 Vol. 133—No. 6

A. W. ALLEN, *Editor*

Comfort of Worker Determines His Efficiency

AIR is essential to active human life. Its chemical composition and physical characteristics affect profoundly the physical or mental efficiency of workers, mental or manual; yet no other subject of vital concern is less understood. Dr. Leonard Hill was one of the pioneers in the collection and dissemination of facts on human reactions to atmospheric conditions. He demonstrated that uncomfortable sensations in the higher ranges of temperature—beyond remedy by the individual in respect to change in clothing—were primarily because of the humidity and stagnation of the air. An experiment of Dr. Hill's demonstrated that discomfort and distress from such conditions can be immediately remedied by circulating the air, thus indicating that the popular demand for fresh air is merely a psychological reaction based on an ignorance of natural cause and physiological effect.

The first practical outcome of Doctor Hill's researches was the invention of the kata thermometer—an instrument that determines the condition of the atmosphere in respect to factors that affect the comfort of human beings at work. By its use a better understanding of the problems involved was gained. Temperature became a matter of less importance than humidity as soon as the fact was realized that surface body moisture must be removed as soon as formed—a transfer best effected by putting the air in movement. The evaporative power of the atmosphere, influenced by all three factors of (1) temperature, (2) humidity, and (3) air circulation, is an indication as to whether or not the prevailing conditions favor the comfort of worker in mine or plant or office.

When temperature is subnormal, atmospheric conditions can be easily remedied to insure comfort. Heat is supplied, and water is added if a deficiency occurs. Air movement within such a temperature range is of secondary importance if the other factors are regulated. When temperature is abnormally high, however, correction is more difficult and more costly, especially when a high humidity also prevails. The temperature of the worker must be maintained uniform by a lower temperature of the atmosphere surrounding him, which is practicable but expensive. This lowering of temperature will automatically free the atmosphere of excess humidity. A less expensive alternative is to encourage evaporation of moisture from the surface of the body by air currents, in which instance comfort depends inversely on temperature and humidity and directly on circulation.

To express atmospheric conditions in respect to comfort in arithmetical terms is difficult. Indexes for manual and sedentary work have been established in connection with the kata thermometer, but the apparatus is comparatively complex, and temperature adjustment to a fixed point must precede each determination. To encourage a broader interest in facts on atmospheric conditions—the influence of air movement and humidity and temperature on comfort, and the effect of comfort on physical and mental efficiency—a description and drawing of a simple indicator apparatus, which can be made by any laboratory worker with a knowledge of glass blowing, will be found on another page of this issue. *Engineering and Mining Journal* will gladly give publicity to research and results in the application of the device to determine the variation from favorable atmospheric conditions in mine or plant or office, for the benefit of the industry as a whole.

Engineering Principles in Accident Prevention

THE BASIC DESIGN of engineering work is an important factor in accident prevention, and any real achievement in safety endeavor is invariably the result of sound planning and the painstaking application of technical knowledge. A case in point is recorded in the 1931 annual report of the Miami Copper Company, wherein Mr. F. W. Maclennan, general manager, states that no fatal accident has occurred in the company's mine in thirty months, during which period 13,287,517 tons of ore was mined by the block-caving system, adopted in 1925. Mr. Maclennan modestly avers that the record speaks well for the skill and judgment of the workmen. It does; but it certainly speaks as well for the skill and judgment of Miami Copper's technical staff. This viewpoint is based primarily on the recollection of a paper by Mr. Maclennan presented at the 1930 annual meeting of the A.I.M.E., in which the author contended that "the greatest contribution a mining company can make toward mine safety is to plan the mining method and the details of operations connected with it in such a way that it is inherently a safe rather than an unsafe operation."

From a safety standpoint, the block-caving mining method employed by Miami Copper would appear to be unusually effective. This conclusion is supported by fatal-accident statistics issued by the United States Bureau of Mines. Those for 1930, the latest available, show that the fatality rate per thousand 300-day work-

ers was 2.52 for mines employing block caving, whereas the rate for all other underground methods was higher, and appreciably so. The rate for top-slicing, for instance, was 3.30; for cut-and-fill it was 3.45; for square-set mining, 3.85; open-stope, 3.76; sublevel caving, 3.96; and shrinkage, 5.98.

The low accident record of Miami Copper may be interpreted as being indicative of advanced thinking in connection with safety. It suggests the application of sound engineering common sense and the basic principles of good business, as well as a humanitarian approach to the subject of work and welfare.

Careful Writing And Poor Engineering

A CASUAL REMARK at the luncheon table of a professional club, by a well-known mining engineer, deserves careful analysis and discussion. It was to the effect that a good writer is never a good engineer.

To define a "good" engineer or a "good" writer is difficult. To compare the inferred quality in each of two individuals is also difficult, because successful engineering connotes knowledge, experience, and capability, whereas effective writing is mostly evidence of a willingness to take pains. A modest engineer-writer, amateur or professional, would disclaim agreement with a verdict that his writing was "good," but he would demur at the assumption that his prospects of success as an engineer would diminish as he attained proficiency as a writer. He can, however, make the incontrovertible statement that many capable engineers are deplorably incompetent writers.

Without repeating the copybook-maxim style of appeal and admonition that characterizes so many treatises on writing in general and technical writing in particular, the profession and industry of mining engineering might well be reminded that the same ideals that govern the design, erection, and operation of mine or surface plant should also govern the work of presenting facts and opinions to form a permanent record. A girder is chosen of a size and shape appropriate to the duty required, with a safety factor of strength that is adequate but not excessive. Words should be used with similar care and frugality, and arranged in a sequence that should appeal to the engineer as logical. A tack used to fasten a filter cloth to a wooden frame is not driven in with a pumpkin; nor should an engineer worthy of the name be satisfied with a written description that fails to convince or inform because of verbosity and pointlessness.

Much of the discourtesy shown to the reading public by careless writers is unintentional. They are convinced, because of long adherence to a philosophy of passivity, that what they call "good" writing is acquired congenitally, instead of being achieved by unremitting self-criticism and repeated revision. Although the average engineer admits the dynamic character of his work —that constant opportunity is presented for improvement by thought and research and experiment—he is inclined to assume that writing is an unimportant detail governed by static rules and regulations. A little interest in the subject, without need to patronize the brotherhood of writers, would convince him that it is susceptible to the laws of logic and science and economics.

Would-be engineers learn, from precept and example, to disregard what is appropriate in word selection and sentence arrangement. The habit thus acquired inhibits the attainment of objective and is contrary to reason. If science, as Huxley averred, is merely organized common sense, scientific principles and scientific practices may be applied with advantage to the art of writing. If the study of economics is of significance, it should be directed toward an appalling waste—the waste of words, materials, and labor, including time wasted by the reader—caused by lack of precision in form and substance of presentation. This aggregate waste can be curtailed or avoided if the writer will consider his reader, and decline to accept the illogical contention that efficiency and effectiveness in the use of language merely constitute evidence of a deficiency in what the complacent engineer might consider as the only essential qualifications for achievement.

Making Alloy Steel At the Mine

STEEL made by smelting a nickel-iron sponge with scrap iron and fluxes in an electric furnace compares favorably with nickel steel of similar composition made by regular methods, Messrs. T. W. Hardy and H. H. Bleakney, metallurgists of the Department of Mines of Canada, report. The process involves the production, by selective flotation, of a nickel concentrate low in copper. Sulphur is reduced to a minimum by roasting, and the resulting nickel-iron oxide is converted into sponge by low-temperature reduction; the gangue is finally removed by magnetic separation, leaving a product with a metallic content of about 90 per cent.

Regular treatment of Sudbury ore involves slagging off much iron to recover nickel and copper—a process that furnishes a constant challenge to the engineer. Like many other problems that are simple at first glance, it also has its complications. By furnishing a concentrate low in copper, the customary objection to the copper content of steel produced directly from ore has lost much of its force. Moreover, much of the prejudice against copper in steel has been removed in recent years. On the other hand, the metallurgist demands metals of higher purity. Alloy manufacture is becoming an exact science, in which the use of virgin metal is desirable, so that accurate composition control may be assured. To maintain a constant ratio between nickel and copper in a natural alloy will not be an easy task.

Selective flotation of nickel-copper ores—an essential preliminary to the new process—has placed the regular reduction method in a more favorable position. A large part of the copper is now recovered as a concentrate, low in nickel, which may be treated in a regular copper smelter; the amount of copper to be separated from nickel by top-and-bottom smelting and by cementation—two costly steps in the refining process—has thus been greatly reduced.

Another objection to the proposed method is the loss of precious metals—an important source of revenue in a refining process by which almost total recovery of

these metals is effected. Regardless of such objections, however, the adoption of selective flotation presents a new angle to the problem, and renewed study is therefore timely. No conclusions regarding the economics of the procedure have been presented, but the results are encouraging, and no technical difficulties are involved. Further reports will be awaited with interest.

At the Parting Of the Ways

AN INITIAL SHOCK to public confidence—the Hatry fraud, a harbinger of world-wide economic disequilibrium—occurred three years ago. Apprehension has since been universal, justified at intervals by disquieting disclosures of the results of overweening and competitive ambition for personal enrichment at the expense of others. Although effort has been made to deflect attention from the fundamental causes of commercial stagnation and social unrest, an occasional tragedy, which hints the existence of countless minor tragedies, claims attention, indicating the futility of hoping that deep-seated economic illness can be cured by strategy and the administration of palliatives and sleeping draughts.

The sooner disagreeable circumstances and facts are disclosed and faced the easier will be the task of moral and material rehabilitation. If knowledge of vital significance to economic progress is to remain hidden, fear will dominate group action in the direction of reform. The best of curative medicines administered by an amateur or a quack may cause physical degeneration and mental decay.

Solution of any problem also necessitates study of the elements that comprise it. Prevailing conditions reflect a loss of confidence in the future among too many people who by their daily purchases and outlook on life can influence national and international stability favorably or unfavorably. Recovery must begin with the individual. From a mass of correspondence from all quarters comes a single piece of private news of an inspiring nature, despite its comparative insignificance: A lad of twenty-one, after the apprenticeship usual in a country where equality and opportunity are not stressed, has been placed on the permanent staff of a large organization. The significant detail in the announcement, however, was contained in the casual intimation that, assuming normal progress, he will be retired on a pension when he reaches the age of sixty, and so removed from the labor market.

Behind this minor fact may be discerned a confidence that engenders stability. The future history of any country can be forecast by assembling items of this character; not by noting the number of plans designed to restore an ephemeral prosperity based on an uncontrolled production program. Nor should too much faith be placed on the ability of any one section of the population to support indefinitely the industries that cater to luxury demand, which is certain to fluctuate, if not seriously to diminish. Sound progress, even though it be slow progress, is more likely to follow the application of a national philosophy that enjoins temperate acquisition and temperate spending.

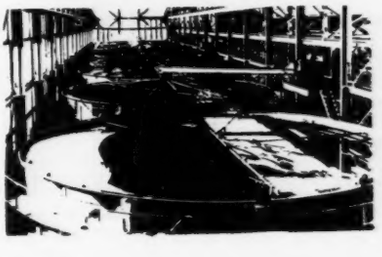
Silicosis Prevention and Increased Drilling Efficiency

IN NEW YORK, recently, was demonstrated the operation of a rock-drill, dust-collecting system devised by Mr. G. S. Kelley, and originally described in the July 13, 1931, issue of *Engineering and Mining Journal*. The device, first used with jackhammers, has now been adapted to stopers and drifters. The demonstration included drilling with all three types. Performance of the collecting equipment was most effective, no dust being perceptible when traps were used.

As pointed out in an article in a recent issue of the *Journal of Industrial Hygiene*, the trap, when used with jackhammers in the drilling of down holes, and when operating at 60 cubic feet of air per minute, will keep the dust concentration at the breathing zone of the drill runners below five million particles per cubic foot, as determined with the Greenburg-Smith impinger. The author of a subsequent article in the same publication, Mr. T. Hatch, describes how use of the trap increases drilling rate:

"During vertical down drilling, without the use of a trap, the dust is blown out of the hole by the air stream that passes through the drill steel and settles on the rock surface at the edge of the hole. As the mound of dust builds up, large particles fall back into the hole and are ground finer before being blown out again. This action produces a continuous cushion of dust between the bit of the drill steel and the rock, and thus retards the drilling speed. When a trap is used, however, the dust is removed as rapidly as it emerges from the hole, and the cushion of dust is therefore reduced, resulting in an increase in the rate of drilling. This is shown by the comparative rates of drilling per hour, with and without the use of the trap. The same drill was used in both tests, and provided with freshly sharpened drill steel. The tests were made within a small area in order to insure constant rock conditions. In order to eliminate the human element in the rate of drilling, a dead weight was applied continuously to the drill (through the courtesy of the Ingersoll-Rand Company a special tripod was provided for this purpose), the drill runner simply operating the machine and steadying it during its operation. Two different weights were used: twenty pounds, which represents the average force applied by the drill runner through his arms, and forty pounds, which is approximately equivalent to the force applied when one leg is thrown over the head of the drill. The results of these tests, the average of two one-hour periods in each instance, show an increase in the rate of drilling when the trap is used of twenty-one and eleven per cent for the twenty- and forty-pound weights, respectively."

From at least two standpoints—prevention of silicosis and increased drilling efficiency—the Kelley dust-collecting system promises much. The apparatus will probably be available as a regular item of manufacturers' equipment in the near future. Keen interest in opportunity for its use on an extensive scale will doubtless be shown by the mining industry, particularly in districts where dust is a menace to health and a hindrance to industrial efficiency.



Processed Crystalline Limestone Finds High-Grade Market

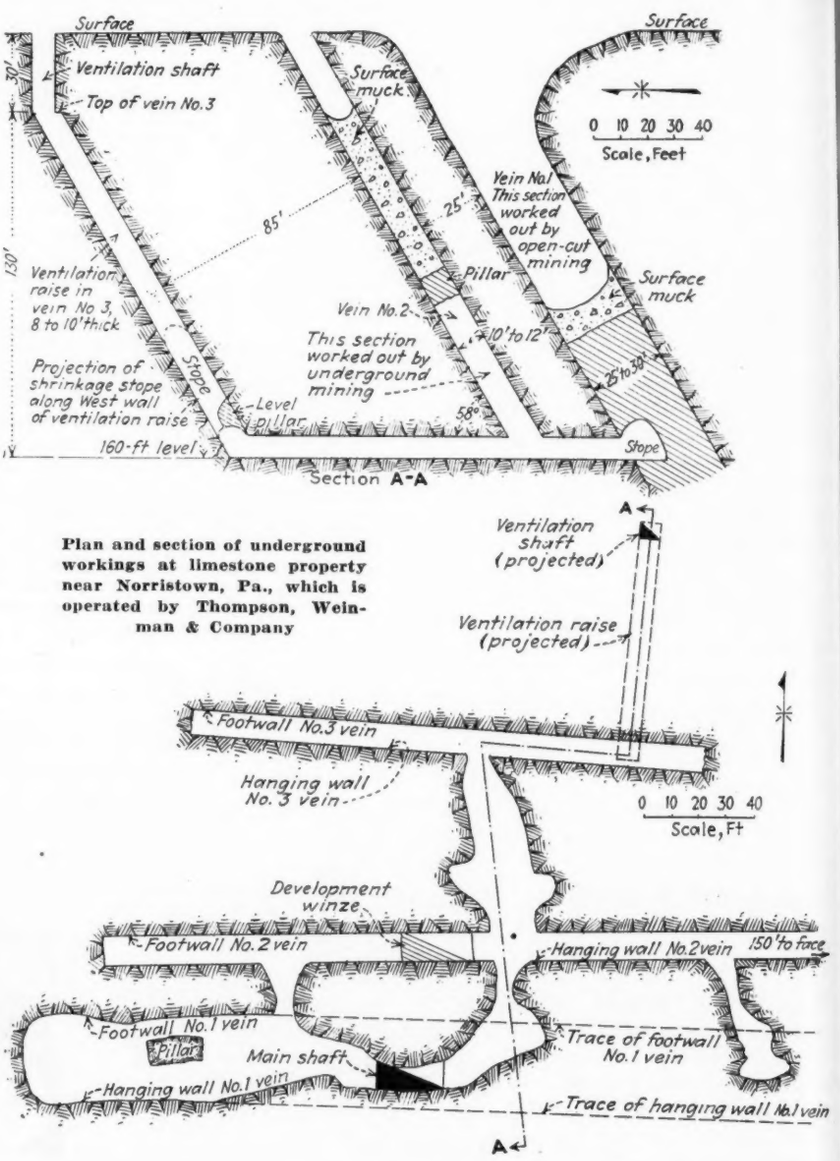
H. D. Keiser
Associate Editor

MINING OF LIMESTONE has been an established industry of Montgomery County, in Pennsylvania, since the early part of the nineteenth century. Extraction methods and processing, however, have undergone numerous changes with the passage of years, principally as a result of economic developments that have periodically affected the industry. Limestone was first used in the county for the walls of farmhouses and barns, and, after it was burned, as farm fertilizer. A fine grade of crystalline limestone, actually a marble capable of taking a high polish, was mined at several properties, and used in the construction of some of the older public buildings in Philadelphia. Subsequent competition offered by marbles from other states eventually made operation of these quarries unprofitable. The limestone was next used for fluxing purposes at local iron furnaces; then, in crushed form, as aggregate for concrete; and, finally, as a source of lime and processed stone for the chemical and manufacturing industries.

One limestone property in the county that has run almost the gamut of the aforesaid developments in utilization of product is that of Thompson, Weinman & Company, near Henderson Station, about three miles southeast of Norristown. The tract, covering about fourteen acres, was acquired in 1928 from the Pennsylvania Railroad, and the mine reopened following many years of idleness. Operations on the property date back beyond the middle of the last century, when it was an important source of building stone. The deposit, as shown in the accompanying sketches, consists of three nearly parallel veins, the strike of which is practically east-west and the dip 58 deg. south. Character of the vein material may be defined[1] as interbedded, coarsely crystalline, white to white and light blue, banded or mottled marble in parallel beds, interbedded with blue to black, hard, siliceous dolomite. The largest and most southern of the three veins, known as No. 1 vein, is 25 to 30 ft. thick, and separated from vein No. 2 by about 25 ft. of siliceous dolomite. Vein No. 2, 10 to 12 ft. thick, is separated by about 85 ft. of dolomite from vein No. 1, which is 8 to 10 ft. thick.

In the early operations on the property, veins Nos. 1 and 2 were worked by open-cut methods. The stone was undoubtedly quarried with plug and feathers, and with wooden plugs, which, after being driven into closely spaced drill holes, were soaked with water, the subsequent swelling of the wood breaking the stone along the line of drill holes.

Vein No. 1 was mined to a vertical depth of about 115 ft., and vein No. 2 to about 90 ft. Operations extended along the strike for several hundred yards. The present operator, in reopening the property, decided to employ underground methods. Two shafts were sunk, and a level was established at a depth of 160 ft. below the surface, as shown in the sketches. The main incline shaft, 10x18 ft. in cross-section, was sunk in vein No. 1 to a point 8 ft. vertically below the level. A 30-ft.



Plan and section of underground workings at limestone property near Norristown, Pa., which is operated by Thompson, Weinman & Company

[1] "Limestones of Pennsylvania," by Benjamin L. Miller, 1932 revision of Bulletin M7, State of Pennsylvania Topographic and Geologic Survey, 1925.

Incline skip road on the footwall of No. 1 vein

shaft, 6x6 ft. in cross-section, was then sunk vertically to the top of No. 3 vein, connecting with a raise driven in the vein from the 160-ft. level. This raise and shaft, known as the ventilation raise and ventilation shaft, are used jointly as an upcast air opening, whereas No. 1 shaft, the main incline shaft, serves as the downcast opening. Ample fresh air for present requirements is furnished by natural ventilation.

A development winze, 10x25 ft. in cross-section, is being sunk in No. 2 vein from the 160-ft. level, at a point about 30 ft. north of the main shaft. When the winze reaches a vertical depth of 320 ft. from the surface, at which depth another level will be established, a crosscut will be driven to vein No. 1, and a raise driven in the vein to connect with and extend the main incline shaft to the new level. Development operations in the three veins will then be started on the level. A vertical depth of 235 ft. has been reached in sinking the winze. At this point, however, a fissure was encountered that contained a large flow of water under hydrostatic pressure. This flow, shown in one of the accompanying illustrations as a stream issuing from a pipe grouted into the drill hole that tapped the fissure, has not as yet been brought entirely under control, and sinking operations have been temporarily suspended. Asphalt grouting methods, developed by the American Asphalt Grouting Company, Chattanooga, Tenn., have been employed in an endeavor to seal off the flow behind the projected walls of the winze. The total amount of water flowing into the mine on the 160-ft. level is about 500 g.p.m. This water is led to a sump, where a 100-hp. centrifugal LeCourtney sinking pump, with a capacity of 1,000 g.p.m. against a head of 250 ft., forces it to waste on the surface. A 100-hp. two-stage centrifugal unit of about the same capacity, made by the Buffalo Steam Pump Company, is available in reserve.

The limestone in veins Nos. 1 and 2 is mined by breast stoping, operations progressing up the dip and on the broken limestone. Stopes about 60 ft. wide are carried up the dip for a distance of about 50 ft., where a continuous horizontal pillar is left between the top of the stope and the accumulation of surface muck in the abandoned open cut above. Pillars between the stopes are spaced irregularly, and range up to 15x15 ft. in cross-section; most of these will eventually be removed following the extraction of all the other ore above the 160-ft. level.

In vein No. 3, which was discovered by the company in crosscutting north from vein No. 2, a shrinkage stope has been opened. This stope, just west of the ventilation raise, is 170 ft. wide along the strike of the vein, and has been carried up the dip for a distance of 45 ft. above the level. Chutes are spaced at 40-ft. intervals. Thickness of the level pillar is a minimum of 20 ft. at the chutes, and a maximum of 35 ft. at the middle point between chutes. The stope is entered through small openings, spaced at regular intervals, that are driven from the ventilation raise through an 8-ft. pillar left between the stope and the raise. Ultimately, when all the limestone above the 160-ft. level has been mined and all the limestone in the new workings off the lower level is mined by the shrinkage method, as is planned, extraction will average about 85 per cent.

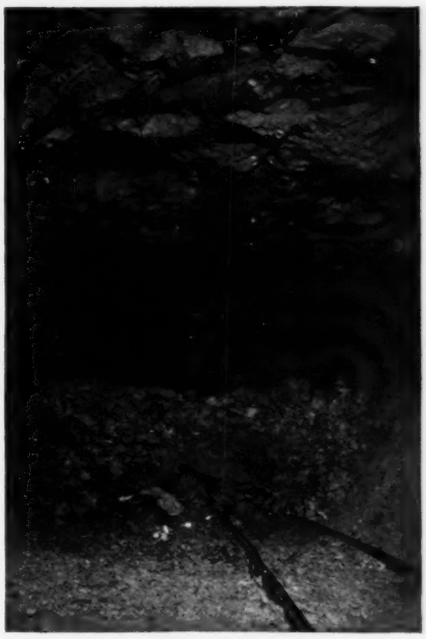
Drill holes are driven 8 ft. deep in both the shrinkage and breast stopes. Spacing of the holes is irregular, and the charges vary, depending on the position of bedding and cleavage planes, which are numerous in the limestone. A 6-ft. "burned" cut, of sixteen 8-ft. holes, is used in drifting. Each hole is charged with five 1¼-in. sticks of Du Pont Gelex No. 2. Ingersoll-Rand drilling equipment is employed: L74 drifters for drifting, S49 stopers and L74 drifters in breast stopes, and S49 stopers in the shrinkage stope.

Detachable bits, made by the Flannery Manufacturing Company, Pittsburgh, Pa., are used with 1-in. round Debitco drill steel throughout the mine. The bits are said to be satisfactory in every respect. One set of four bits, used with a set of 2-, 4-, 6-, and 8-ft. drills, is reported still in use after drilling a total of 1,300 ft. in the limestone without resharpening. The bits are stocked by a local supply house and requisitioned as required.

Limestone mined in the breast stopes is loaded by hand into end-dump, 14-cu.ft. cars, which hold about a ton. The cars are trammed by hand over 20-lb. rails, with a gage of 24 in., to the main shaft, where the limestone is dumped directly into a 1-ton skip. Limestone mined in the shrinkage stope is drawn through chutes into the same type of car and trammed in the same manner to the main shaft. The skip has a gage of 54 in., and is pulled up a 250-ft. incline—180 ft. from the 160-ft. level to the "knuckle" at the surface, and 70 ft. from the knuckle to the mill—by a 40-hp. single-drum hoist. From the skip the limestone is dumped into a No. 6 McCully gyratory crusher, the 4-in. product of which passes to a 12-in. bucket elevator that discharges into three 60-ton mill feed bins.

On leaving the mill feed bins, the

Flow of water encountered in development winze



East development drift in No. 3 vein on 160-ft. level

limestone is processed by one of two systems that are employed: a dry system and a wet system. In the dry system the limestone is ground in four-roller, high-side Raymond mills. Two products of different size are made. One is 98 per cent through 200 mesh, and the other 99 per cent through 300 mesh. Both are packed in paper bags for shipment. In the wet system the limestone undergoes wet grinding in Hardinge mills and subsequent hydraulic classification. Discharge from the classifiers is settled in Dorr thickeners, and the settled product is dried on Cartersville steam-drum dryers. The dried material, 100 per cent minus-325 mesh, is packed in paper bags. St. Regis and Bates Valve bag-packing machines are used. Periodic samples are taken daily at the bag-packing machines. These samples are tested on the property for fineness and color.

The quantity of limestone mined and milled averages normally about 2,000 tons a month. The underground force consists of nine men: two drillers, two helpers, and five muckers. Mucking is done on a day shift of ten hours, and drilling on two shifts: a day shift of ten hours and a night shift of the same length. One stoper and a drifter are operated on each drilling shift. A six-day working week is in effect. Of the total tonnage mined, only 1 per cent is waste. This is sorted out underground, and later hoisted separately and transported over a trestle that extends from the top of the incline to a waste pile, where the material is dumped.

The mill has a capacity of 2,000 tons per month, on a 24-hr. operating basis for the wet system and a 10-hr. basis for the dry system. Output of the mine, however, with existing facilities, can be increased to 3,000 tons per month on a 10-hr. basis; that is, 100 tons can be mined and hoisted on each 10-hr. shift, if desired. The mill crew consists of nine men: three mill operators on 12-hr. day and night shifts; one fireman, one dryer-man, and four packers on a 10-hr. day shift. Electric power for the mill is obtained from the lines of the Philadelphia Electric Company. In addition to the mill building, the surface plant includes a 50x80 ft. dryer room; 40x50 ft. boiler house, where steam for the dryers is produced; and a 14x30 ft. office and supply building.

The three aforesaid products made in the mill are used by rubber, paint, putty, and linoleum manufacturers. Commercially, the product sized to 98 per cent through 200 mesh is known as Norris Filler; that 99 per cent through 300 mesh as Norris White; and the wet ground product, 100 per cent through 325 mesh, as Keystone Paris White. Besides these products an admixture for concrete is prepared. This has been given the trade name of Tensilite. All the products, after being packed in bags, are loaded into box cars for shipment. The siding alongside the mill connects with the main line of the Pennsylvania Railroad at Rambo, Pa., 2½ miles east of the property.

Permission to visit the Norristown, Pa., operations of Thompson, Weinman & Company, and to prepare the foregoing account, was granted by S. W. Thompson, sales manager of the company. Descriptive material and drawings were supplied by F. Chappell, the company's superintendent at Norristown. For the historical facts presented, in addition to the geologic description of the deposit, acknowledgement for which has already been made, I am indebted to Dr. Benjamin L. Miller, head of the department of geology, Lehigh University, who courteously permitted me to read and draw from his manuscript for the revised edition of "Limestones of Pennsylvania," soon to be published.

■ ■ ■

Motor-Truck Drilling Platform Reduces Tunneling Costs

HOLES for blasting almost the entire face of a 56-ft. tunnel were all drilled simultaneously at the Hoover Dam, with the aid of a motor-truck platform, shown in the accompanying illustration, thereby effecting economies in time and labor. The ground stands well, and no timbering is required. A timber framework mounted on wheels is used as a platform for the daily inspection and barring down of the back.

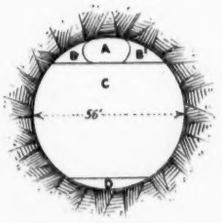
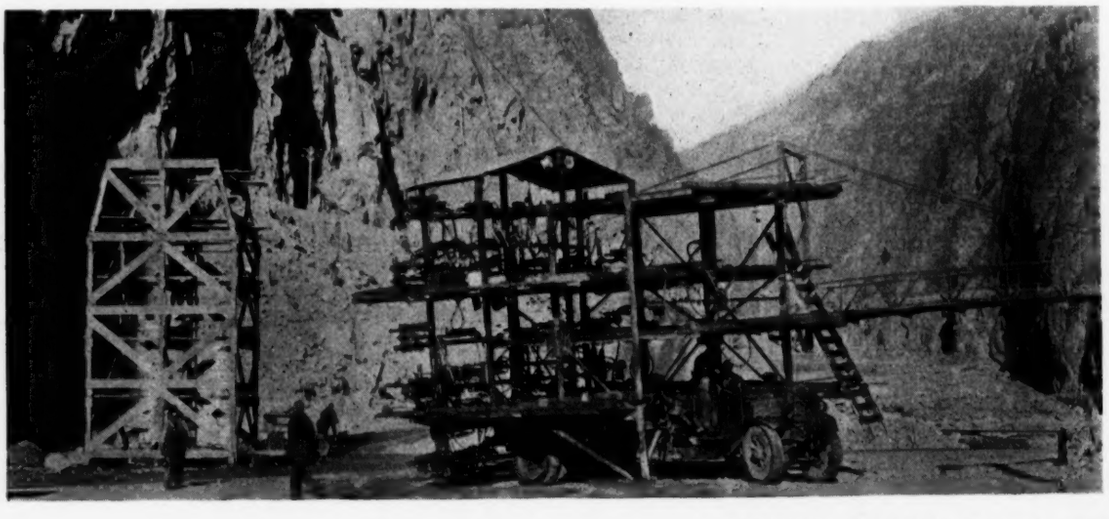


Diagram of mining sequence

■

A description of drilling practice will be simplified by reference to the accompanying drawing. The original small heading is shown at *A*, and the ground on each side (*B* and *B'*) is blasted out before multiple drilling (16-ft. rounds) is commenced over the larger area *C*, using the drilling platform. The bottom space *D* is left until after concreting, which is not commenced until the entire tunnel, less the small segment at the bottom, has been completed.

Drilling and inspection platforms



Amalgamating and Cyaniding on Siscoe Island, Quebec

C. O. Stee

By means of a simple arrangement at the discharge end of the ball mills, followed by amalgamation, 78.5 per cent of the gold in the ore is recovered. Blankets in the circuit recover concentrate that yields an additional 7.0 per cent. Cyanidation recovers 13.0 per cent. Thus, a total recovery of 98.5 per cent is made. These are some of the interesting accomplishments at the mill described in this article.

THE Siscoe Gold Mines property is situated on and surrounds Siscoe Island, of about 340 acres, in Lake Dubuisson, County of Abitibi, Province of Quebec, Canada. The nearest railway point is Amos, 43 miles to the northwest, on the Canadian National main line from Quebec to Cochrane and west. From Amos to the mines is an excellent waterway up the Harricana River and through a series of large lakes. Passenger and freight steamers make trips on regular schedule during the navigating season. Throughout the year the planes of General Airways fly the mails and also carry passengers, and excellent service is maintained. Noranda Mines is 53 miles almost due west. Sixteen miles to the east is the recently discovered Pascallis gold district.

Gold was first discovered on Siscoe Island by Stanley Siscoe, now a director of the company, in 1913. Two years previously the first gold discovery in the district was made in a small quartz vein on the mainland of the lake, about a mile southeast of the island, by J. J. Sullivan and Hector Authier, on the property now owned by Sullivan Gold Mines. Followed many years of struggle to finance and develop the Siscoe property, looked upon by many as a wildcat. In fact, until recently, well-known engineers who examined the property reserved favorable opinion. But the untiring efforts of John T. Tebbutt and Stanley Siscoe kept it alive.

Four shafts, *A*, *B*, *C*, and *D*, were sunk on various showings. The first two were shallow, and no lateral work was done from them. They were abandoned. The *C* shaft, an incline, was sunk on a vein that outcropped for a distance of about 300 ft. From this shaft, drifts were made on the vein on three levels, at intervals of 165 ft., 282 ft., and 390 ft., respectively. The *D* shaft, also an incline, was sunk to a depth of 125 ft., on another showing. From this, considerable lateral work was done. In the workings from these two shafts some spectacular gold specimens were found.

About this time the present company was formed. In July, 1928, J. M. Forbes, the present general manager, was sent by the brokerage firm of Stobie-Forlong to examine the property; and, on the strength of his recommendation, financing by Stobie-Forlong was undertaken. This was easily accomplished. A mill and power plant were completed in the fall of 1928. In January, 1929, the property went into production.

The Siscoe mine is in a granodiorite intrusive near the contact with greenstone. This contact crosses the north half of Siscoe island, and has a northwest strike. Some minor veins have been discovered in the greenstone to the south of the contact, but the workable veins have so far been found in the granodiorite on the north side. These veins have a strike that varies from 60 to 90 deg. to the contact. The *C* vein has a dip of 38 to 45 deg.; the veins in the Western section, on the contrary, dip to the east 45 to 65 deg. Except in one occurrence, where a small lens was found in the contact zone, the main oreshoots are not within 250 ft. of the contact. The veins mined to date consist of white quartz, and vary in thickness from a few inches to 12 ft. In the quartz are found streaks and bands of black tourmaline, and frequently impregnations of pyrite and chalcopyrite. The presence of sulphide, pyrite and chalcopyrite is usually accompanied by high values. The gold occurs in the free state and at times can be seen in spectacular display.

To the northwest of the present workings is the Siscoe vein, now being developed. This differs from the others in that the quartz is a bluish gray. The mineralized area is considerably fractured and faulted. Along some of these faults, displacement may be as much as 25 ft. Through the heart of the mining area, parallel to the veins and having a vertical dip, is an albite dike about 15 ft. thick, which cuts the veins and all other formations, and is therefore of a later date. We do not know whether it passes through the contact into the adjoining greenstone or not.

Another series of dikes cuts the veins at various angles, classified as diorite dikes, varying in thickness from a few inches to 2 ft. The walls are usually sharply defined, and only in rare instances have values been found in the granodiorite, and then when in close proximity to quartz. The granodiorite in the Siscoe vicinity is of a much finer texture than that in the larger part of the intrusive area to the east. Doubt exists as to whether the Siscoe area is a tongue protruding to the west from the main body of granodiorite, or is an island completely surrounded by greenstone. Only extensive diamond drilling in the surrounding lake can solve this problem.

The ore is mined by shrinkage, hand trammed to the shaft, and hoisted by caging the cars. The main shaft has three 4½x5-ft. compartments. The bottom level is 600 ft. below the collar. This shaft is now being deepened to 870 ft. Cages operate in counterbalance. Cars are emptied into an ore bin through a grizzly made of heavy railroad rails with 8-in. spacing.

The mine ore bin, with a capacity of 125 tons, empties through an open gate unto a 24x9-in., ratchet-driven apron feeder, which discharges to a fixed-bar grizzly (25x43 in.) set at 45 deg. and having 1½-in. spacing between bars. Oversize passes directly to a 15x20 Universal jaw crusher, 1½-in. opening, with a capacity of 20 tons per hour. It is driven by a 35-hp. CGE induction motor at a speed of 200 r.p.m. The starter is of the drum-controller type, preceeded by a square *D* and oil switch. A set of jaw plates is good for about 10,000 tons. Cheek plates will last 15,000 tons. The crushed ore joins the undersize from the grizzly and passes to a 14-in. by 100-ft. belt conveyor, set at 20 deg., which discharges to the rope tramway loading bins, which have a capacity of 150 tons.

Central shaft is about 800 ft. from the mill. Until recently all ore was hoisted through the old incline *C* shaft direct to the bins of another crushing plant near the mill. For various reasons this was not satisfactory. When

Surface plant at the Central shaft

the decision was made to hoist all ore through Central shaft, the question arose as to what method should be employed in the transfer of the ore from this shaft to the mill. A belt conveyor was considered, and complete estimates on cost were worked ou.. As an alternative, a rope tramway was investigated, and selected in preference to the conveyor-belt method. This tramway, of the twin-rope shuttle type, with a single bucket and one man operating, was supplied by the Interstate Equipment Corporation. The carrier ropes are of smooth-surface lock coil 1¼-in. diameter. The tractor rope is ½-in., cast-steel 6/19 wires.

Driving machinery consists of a 10-hp. continuous-rating induction motor, having a drum controller type starter. This is direct-connected through reduction gear, which in turn drives the clutch wheel. On the shaft of the latter is a friction band brake of large diameter, by means of which the bucket is always under control of the operator through a brake lever. Brake lever and controller are conveniently arranged on the loading platform. The bucket is of hopper-bottom type, with a door on the front side that locks by means of bolt and spring. This bolt can be pulled, and the load dumped by means of an automatic device at any point along the line, or by a tripping lever at the discharge terminal. The bucket has a capacity of 2 tons of ore, and rides on four double-flanged, 12-in. diameter wheels, equipped with roller bearings. It is loaded through steel chutes equipped with counterweighted undercut gates. Traction rope speed is 600 ft. per minute. A complete round trip is made every three minutes. The capacity of the system is 40 tons per hour. One man does all the work. Cost of transferring ore by this method is 5c. per ton.

The tramway bucket discharges into the mill bins, which have a capacity of 350 tons. From these bins the ore passes over two Southwestern belt-type ore feeders to Marcy ball mills, equipped with combination external and internal scoop feeders. One of these mills is Size 64½ and the other Size 55. The first will grind up to 120 tons per day, has a speed of 26 r.p.m., and is driven by a 100-hp. induction motor at 690 r.p.m. through a Texrope drive with 54-in. centers. The other has a capacity of 60 tons per day, a speed of 29 r.p.m., and requires a 50-hp. induction motor running 1,160 r.p.m., with Texrope drive. Both are equipped with drum controller type starters and the necessary switches and oil-circuit breakers. Forged chrome 5-in. steel balls are used. Ball charges are 11,000 and 6,000 lb., respectively. Ball consumption for the past ten months has been at a rate of 2.22 lb. per ton for the larger mill, and 2.26 lb. per ton for the smaller. With the mills operating about 94 per cent of the time, a set of liners will last about eight months. Feed and shell liners are of manganese steel, and the discharge grates are of high-carbon chrome steel. The ball-mill discharge averages 80 per cent solids.

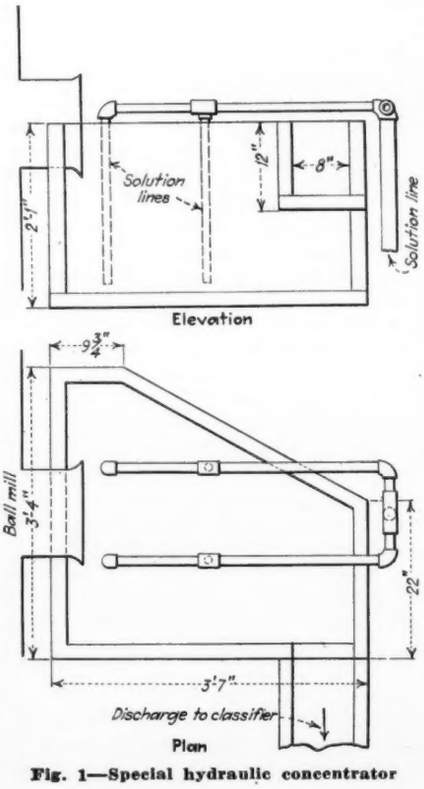


Fig. 1—Special hydraulic concentrator

An increase in mill capacity, without adding additional ball mills, has been suggested as practicable by adopting finer secondary crushing. Some assert that as much as 50 per cent increase could be expected if the ball-mill feed was reduced from the 2-in. maximum, as at present, to a ½-in. maximum. To test this theory, a screen of the Niagara type was placed at the end of the conveyor belt at the top of the tramway loading bins. These bins were divided so that the plus ½-in. material would go to one, and the minus ½-in. material to the other. At the mill bins was a similar division. This made possible the delivery of coarser product to one ball mill and finer to the other. The minus ½-in. was fed to No. 2 ball mill, which, as previously stated, is a Size 55 Marcy. The test extended over two months, during which time every effort was made to increase tonnage. The circulating load, with increased feed, quickly accumulated until the mill was receiving more than it would handle. During the first month, 2½-in. balls were used. These gave a finer product but no increased tonnage.

Table I—Ball-Mill Tests

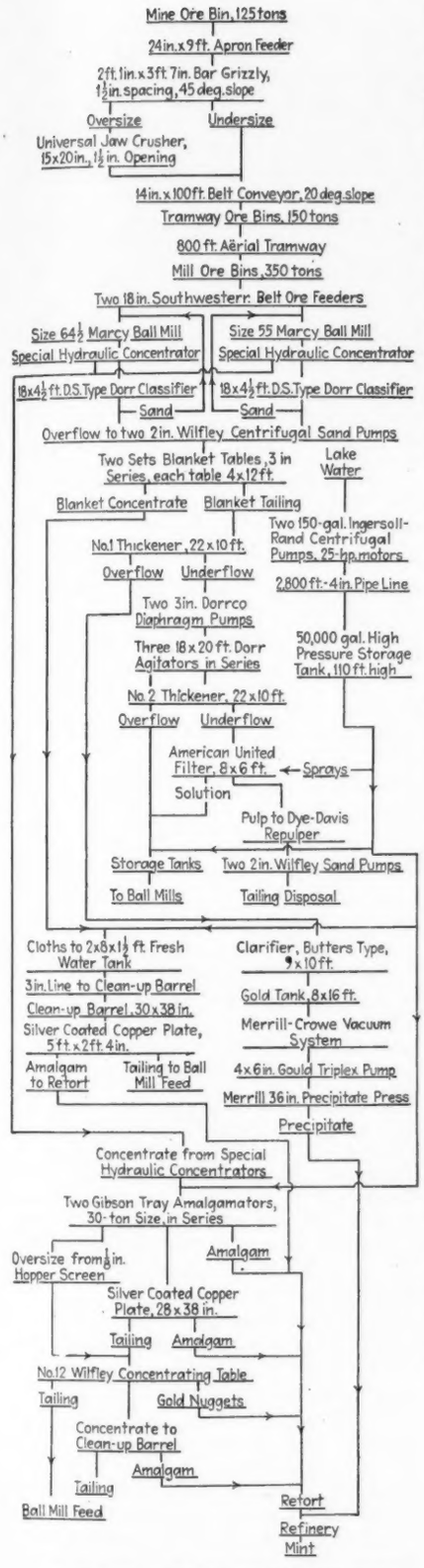
	Ball Mill No. 1, Tons per Hour	Ball Mill No. 2, Tons per Hour
Month of November preceding test	5.17	2.556
Month of December during test	5.16	2.553
Month of January during test	5.21	2.530

During the second month, 5-in. balls were used. The tonnage remained about the same, with a slightly coarser product. Table I gives the average tonnage handled by each mill per hour during the month preceding the test and during the two months of the test.

If the discharge from the ball mills went to some other mill for finer grinding, instead of to a classifier in closed circuit, the mills would handle a much larger tonnage of the finer feed. But the road to increased tonnage is not by way of finer crushing, but rather by installation of more ball mills of the type being used. Screen tests are given in Table II.

In close circuit with each mill is an 18x4½-ft. D.S. Dorr classifier, set at an inclination of 2.26 in. per foot. The circulating load to No. 1 mill is about 13 tons per hour, and that to No. 2 is 8 tons per hour. Ratio of classifier return to mill feed is 2.72 for the larger mill, and 3.15 for the smaller mill. The life of a set of rakes is about 85,000 tons discharged to the mill. Moisture content of circulating load is 17 per cent. Rakes make 27 strokes per minute.

Except for a small percentage contained in the sulphides, the gold in the Siscoe ores is in a free state and often coarse. Nuggets have been discharged from the ball mills that weighed as much as one ounce apiece. Obviously, this gold would not overflow the classifier, nor could it be recovered by


Mine Ore Bin, 125 tons
24 in. x 9 ft. Apron Feeder
2 ft. 1 in. x 3 ft. 7 in. Bar Grizzly, 1½ in. spacing, 45 deg. slope
Oversize
Undersize
Universal Jaw Crusher, 15x20 in., 1½ in. Opening
14 in. x 100 ft. Belt Conveyor, 20 deg. slope
Tramway Ore Bins, 150 tons
800 ft. Aërial Tramway
Mill Ore Bins, 350 tons
Two 18 in. Southwestern Belt Ore Feeders
Size 64½ Marcy Ball Mill Special Hydraulic Concentrator
Size 55 Marcy Ball Mill Special Hydraulic Concentrator
18x4½ ft. D.S. Type Dorr Classifier
18x4½ ft. D.S. Type Dorr Classifier
Sand
Sand
Overflow to two 2 in. Wilfley Centrifugal Sand Pumps
Two Sets Blanket Tables, 3 in Series, each table 4x12 ft.
Lake Water
Blanket Concentrate
Blanket Tailing
No. 1 Thickener, 22 x 10 ft.
Two 150-gal. Ingersoll-Rand Centrifugal Pumps, 25-hp. motors
Overflow
Underflow
2,800 ft.-4 in. Pipe Line
Two 3 in. Dorrco Diaphragm Pumps
Three 18 x 20 ft. Dorr Agitators in Series
50,000 gal. High Pressure Storage Tank, 110 ft. high
No. 2 Thickener, 22 x 10 ft.
Overflow
Underflow
American United Filter, 8 x 6 ft.
Sprays
Solution
Pulp to Dye-Davis Repulper
Storage Tanks
Two 2 in. Wilfley Sand Pumps
To Ball Mills
Tailing Disposal
Cloths to 2x8x1½ ft. Fresh Water Tank
Clarifier, Butters Type, 9 x 10 ft.
3 in. Line to Clean-up Barrel
Gold Tank, 8 x 16 ft.
Clean-up Barrel, 30 x 38 in.
Merrill-Crowe Vacuum System
Silver Coated Copper Plate, 5 ft. x 2 ft. 4 in.
Amalgam to Retort
Tailing to Ball Mill Feed
4 x 6 in. Gould Triplex Pump
Merrill 36 in. Precipitate Press
Precipitate
Concentrate from Special Hydraulic Concentrators
Two Gibson Tray Amalgamators, 30-ton Size, in Series
Oversize from ⅛ in. Hopper Screen
Amalgam
Silver Coated Copper Plate, 28 x 38 in.
Tailing
Amalgam
No. 12 Wilfley Concentrating Table
Tailing
Gold Nuggets
Concentrate to Clean-up Barrel
Amalgam
Tailing
Ball Mill Feed
Retort
Refinery
Mint

Flowsheet of Siscoe Gold Mines' mill

cyanidation if it did. To recover this in a concentrate and treat it later by amalgamation, a special box-type hydraulic concentrator was designed, placed at the discharge end of each ball mill, ahead of the classifier. This box, as illustrated in Fig. 1, has four 1-in. pipes, placed vertically, the open ends being 1 in. from the bottom. These pipes are connected to the mill solution lines. Solution under pressure keeps the sludge in the box, as it comes from the ball mill, well stirred up. The lighter materials overflow into the classifier while the heavier particles, including the free gold, remain in the box. At the end of each 8-hr. shift the mill is shut down for a few minutes. the solution cut off, and the pipes are turned out of the way. A man removes the concentrate with a square-pointed shovel, placing it in tubs, which are later emptied into a feed box at the amalgamating unit. Amount of concentrate averages about 1.75 tons per day, or about 1 per cent of the mill feed. It contains 78.5 per cent of the gold in the ore.

The amalgamating unit consists of two Gibson tray amalgamators, in series, on a timber frame; followed by a silver-coated copper plate, 28x38 in., and a No. 12 Wilfley concentrating table. The frame of the Gibson amalgamators rests on two 3-in. pipes, with strap-iron wearing plates above and below. The 3-in. pipes act as rollers on which the unit can oscillate back and forth freely. A standard No. 6 Wilfley improved self-piling head motion is used—a satisfactory arrangement.

A special type of hopper with double-screen bottom has been provided for the upper unit of the Gibson amalgamators. The first screen has ¼-in. openings; the next, ⅛-in. openings. The first screen prevents tramp iron and coarse material from reaching the amalgamator and clogging the compartments. Occasionally, a large nugget is picked from this screen. Oversize from the second screen, which also contains coarse gold, passes through a pipe direct to the Wilfley table. Here the coarse gold is picked off the table, as it separates out, by an operator stationed at the end of the table. The material that passes the screens is the feed to Gibson amalgamators.

Clean water, from a series of sprays placed over the hopper, washes the concentrate through the machines. Mercury is fed directly into the first compartments of the amalgamator; consumption is 1 oz. to every 12 oz. gold. To keep the mercury in condition, about ¼ lb. of sodium hydroxide is consumed per day. The concentrate is fed to the hopper gradually by hand. About four hours are required to pass the concentrate through the amalgamator.

Aside from the free gold picked from the hopper and off the Wilfley table, the major portion combines with the mercury and is lodged in the first

Table II—Screen Tests, Ball-Mill Feed

Mesh		Ball Mill No. 1, %	Ball Mill No. 2, %
On	4	100	45.80
	10	100	20.40
	20	100	13.36
	40	100	7.72
	60	100	2.74
	80	100	0.65
	100	100	1.45
	150	100	1.36
	200	100	1.03
Minus	200	100	5.18

Ball-Mill Discharge

Mesh		Ball Mill No. 1, %	Ball Mill No. 2, %
On	4	4.43	3.10
	10	4.71	1.80
	20	9.75	4.55
	40	25.80	17.31
	60	18.53	19.76
	80	6.25	12.85
	150	12.20	9.24
	200	3.32	5.98
Minus	200	14.58	17.47

Classifier Return

Mesh		Ball Mill No. 1, %	Ball Mill No. 2, %
On	10	5.37	7.10
	20	10.58	6.96
	40	32.72	24.44
	60	25.08	25.44
	80	7.03	8.10
	100	8.65	13.86
	150	3.82	7.38
	200	2.55	2.94
Minus	200	3.49	3.28

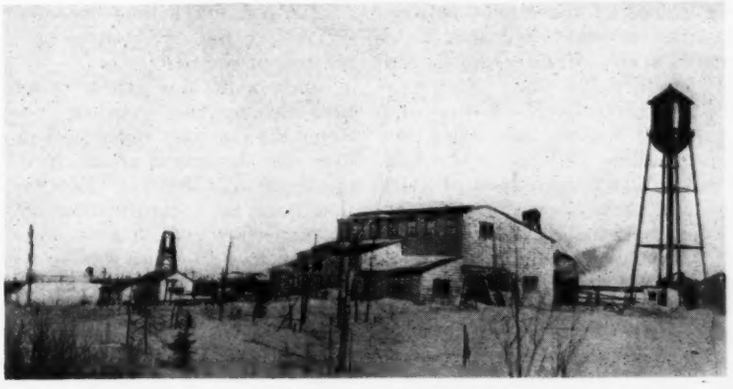
Classifier Overflow

Mesh		Classifier No. 1, %	Classifier No. 2
On	60	13.43	1.09
	80	4.59	1.48
	100	12.62	7.46
	150	9.95	14.10
	200	9.75	13.39
Minus	200	49.37	62.04

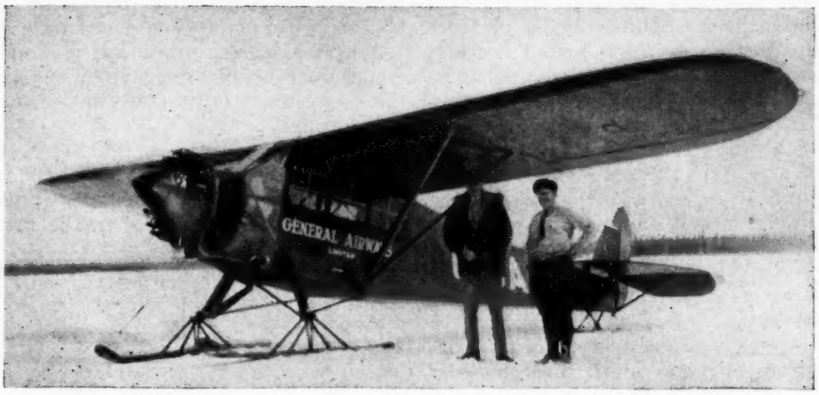
Note: Overflow from classifier No. 2 represents pulp to cyanide plant.

Table III—Metallurgical Results, Oct., 1931-March, 1932, Inclusive

	Tons Treated	Recovery Per Ton Milled	Recovery Per Cent
Amalgamation	31,497	$13.308	86.125
Cyanidation	31,497	1.921	12.432
Totals	.....	$15.229	98.557
Tailing	.....	0.223	1.443
Grade of ore	.....	$15.452	100.000



Mill of Siscoe Gold Mines



Regular airplane service connects Siscoe with all parts of northern Quebec and Ontario

few compartments of the upper unit. A small amount of amalgam finds its way into the compartments of the lower unit and even onto the plate that follows. At the end of the plate and just ahead of the table is a mercury trap. In the course of a few days this trap accumulates a small quantity of amalgam. Tailing from the table is returned to the ball-mill feed. Concentrate is put through a clean-up barrel. If it does not appear rich, it is also returned to the ball mills.

When the day's accumulation of concentrate has been run through, the trays are thoroughly washed by a stream of water for about 15 min., after which the covers are removed and the compartments cleaned. The best tools for this are a square-pointed blade and a small spoon with the handle bent into a vertical position. After the compartments are cleaned, the amalgam is dumped on the silver plate, and sand and small pebbles are washed out by means of a stream of water from a small hose, while the amalgam is being worked over by hand. The amalgam is then retorted in the usual way. The resulting gold chunk, averaging $17.50 to $18 per ounce, is further refined before shipment to the mint.

This, however, is not all the gold recovered by amalgamation. At times the mill tailing contained higher values than the average. The assumption was that relatively coarse gold was finding its way into the cyanide unit and was incompletely dissolved. To overcome this, we placed in the circuit between the classifier overflow and the No. 1 thickener a series of three tables with blankets on them, in line. Each is 4 ft. wide and 12 ft. long. A drop of 6 in. is provided between each table, and they have a slope of 1 in. to the foot. The blanket material consists of thick silence cloth, such as used on the average dining-room table. This cloth is laid on the tables in lengths convenient to handle—about 4 ft. The upper pieces overlap those below. No fastening is necessary. At the upper end of the table a distributing box is provided. The pulp from the classifier overflow is pumped to this box through 2-in. pipe lines from 2-in. Wilfley and Southwestern sand pumps. As the pulp flows evenly over the surface of the tables, the larger gold particles settle and adhere to the cloth. Once each shift the cloths are removed and washed in a tank of water. When the material has settled, the water is siphoned off, and the pulp run by gravity from the bottom of the tank through a 3-in. pipe line to a 30x38-in. clean-up barrel. Water and lime are added, and the barrel is rotated at 30 r.p.m. for six hours. The barrel is then stopped and about 25 oz. of mercury added. It is then run for two hours more. A plug in the shell of the barrel is then removed, and the contents are allowed to discharge onto a silver-coated copper plate (5 ft. by 2 ft. 4 in.) set on a slope of ¾ in. to the foot. At the bottom end of this plate is an amalgam trap. The amalgam is captured by the plate and the pulp overflows into a tank, from which it is returned to the ball-mill feed. The amalgam on the plate is cleaned up in the usual way and retorted. From this portion of our flowsheet we recover an additional 6 to 7 per cent of the gold in the ore. Only 12 to 14 per cent remains for the cyanide unit.

The cyanide method employed is simple. Grinding is done in cyanide solution. Lime and cyanide are added at the ball-mill intake. The grinding solution is kept at 0.9 lb. (NaCN equivalent), and other solutions are maintained at 0.8 lb. per ton of solution. Cyanide consumption is 1.08 lb. per ton of ore. Average lime consumption is 0.7 lb. per ton of solution, or 1.36 lb. per ton of ore.

Much of the fine gold goes into solution during the grinding operation. After the classifier overflow has passed over the blankets it enters No. 1 Dorr thickener, 22x10 ft. The clear solution goes to a clarification tank, and the underflow, which averages 0.75 of solution to 1 of solids and is continued at this figure throughout agitation, is pumped by two 3-in. Dorrco diaphragm pumps to No. 1 Dorr agitator. There are three agitators, each 20x18 ft., in series. Time of agitation is about 44 hours: 130 cu.ft. of compressed air per minute at 40-lb. pressure is required for circulation. From No. 3 agitator the pulp passes to No. 2 thickener, which is also 20x18 ft. Clear solution from this goes to storage. The pulp, which contains 0.5 solution to 1 of solids, is elevated by two 3-in. Dorrco diaphragm pumps and sent to an 8x6-ft. American filter, capacity of which is 48 lb. per square foot per hour. The life of the filter bags is about three months. In this filter, 25 to 28 in. of vacuum is maintained. Solution drawn off is returned to storage. The filter cake is washed with a barren solution and water, and discharged into a Dye-Davis cake repulper, where water is added. The pulp then passes to a 2-in. Wilfley centrifugal sand pump and is pumped to tailing disposal.

To return to the place in the flowsheet occupied by the No. 1 thickener: the overflow clear solution from this is sent to a Butters-type clarifier, where the final sediment is removed. The clarifier tank is 9x10 ft., and the leaves are 5 ft. 5 in. by 4 ft. 4 in. Clarified solution is pumped by a double-suction volute Class L. V. Cameron pump to a 15x8-ft. gold solution tank. The Merrill-Crowe vacuum system of precipitation is employed, with a 4x12-ft. receiver, the solution being circulated by a Gould 4x6 triplex pump. Immediately before the solution enters this pump, zinc dust is added, delivered from the zinc feeder into a mixing cone into which a small stream of solution from the discharge of the triplex pump is kept flowing. One pound of lead acetate is added by drip into the solution every 24 hr. The mixture of zinc dust and solution passes through a 1-in. line to the suction of the pump. Zinc consumption over a ten-month period has averaged 0.05 lb. per ton of solution. Precipitate is recovered in a 36-in. Merrill precipitation press, Type C, which is cleaned up twice each month. At the end of the month the precipitate is mixed with fluxes, melted into a gold button, and refined with the accumulated retort gold.

To operate a complete cyanide unit as described in the foregoing for the recovery of only 12 to 14 per cent of the values in the ore would seem like an inefficient procedure. However, the cyanide plant is returning a good profit on its operation. Another method, however, might be substituted. Some experimental work has been done with flotation of the classifier overflow, the cyanide feed, and a 25 to 1 concentrate can possibly be made which could be treated by cyanidation in a smaller unit with a material saving in operating expense. However, the recovery by the method here described is averaging over 98 per cent of the total gold in the ore.

For assistance in the preparation of this article, grateful acknowledgment is made to the mill superintendent, Owen Matthews, in the gathering of data on the operation of the plant.

Cost of Shrinkage Mining

Morris J. Elsing

SHRINKAGE MINING shows a wider range in costs per ton than any other method. An orebody must possess numerous characteristics in order that it may be successfully mined by shrinkage, but the cost per ton depends largely on size. The method is applicable to large, thick deposits or narrow veins that possess the other necessary requirements for shrinkage mining. In the first instance low costs are obtained; in the latter, the costs are high. Between these costs is an extremely wide range.

According to the U. S. Bureau of Mines, approximately 8,000,000 tons of ore was mined in 1929 in the United States by this method. A considerable amount was produced by mines that use shrinkage as an auxiliary to caving. Some mines use shrinkage, cut-and-fill, and the square-set method. The published cost in such event is not representative of any particular method of mining. Representative costs of mining by shrinkage are obtainable only by reference to the many mines where this is the only method employed.

Shrinkage mines have been divided into three classes: narrow veins; wide veins and medium-large oreshoots, and large deposits. The first to be discussed are the costs of mining narrow veins, which vary in width from 4.5 to 12 ft., shown in Table I.

Nevada-Massachusetts has a daily production of 142 tons of scheelite ore, which occurs in beds in limestone having a dip of 75 deg. The bedded walls stand well with moderate support, and the ore breaks easily. The mine is developed by incline shafts. Harmony mine has a daily production of 150 tons. The vein dips at 65 deg., and it has hard, strong shale walls. The ore is chalcopyrite in a siliceous gangue, which is hard to break. This mine is developed by tunnels and a 150-ft. winze.

Cortez Silver has a daily production of 125 tons of lead-silver ore. The veins dip from 45 to 80 deg. They have poorly defined walls, which are solid and stand well. Wall rocks are limestone and quartzite. The mine is developed by a tunnel and an incline shaft 240 ft. below it.

The Mogollon mine has a daily production of 150 tons. The width of the vein in the larger stopes is from 8 to 15 ft. In minor veins the width is from 2.5 to 4 ft. Average width has been taken as the average of these widths. The gangue, which is hard, is quartz, calcite, and andesite breccia.

This article is the seventh in a series on the cost of mining in the United States.

The mine is 75 miles from the railroad and the cost of transportation is 64c. per ton of ore milled, part of which is reflected in the cost of mining.

Smuggler Union, when operating in 1927, produced about 700 tons per day. The Smuggler vein is 3 ft. wide and dips at 79 deg. The Flat vein is 4 ft. wide and dips at 60 deg. Costs, shown in Table I, apply to ore from a portion of the combined veins, which has a width of 10 ft. The ore is gold and silver with a little lead. Stopes were 225 to 250 ft. long with pillars 20 ft. wide. The veins occur in andesite and breccia.

Rosiclare mine has a daily production of about 170 tons. The vein dips steeply, and the oreshoot is continuous for 1,900 ft. in length and to the bottom of the mine, which is at the 550 ft. level. The walls are solid limestone except in the upper level, where mining is done with square-sets. Shrinkage stopes produce 75 per cent of the tonnage, and 25 per cent is produced by square-set stopes. Most of the timber reported is used in the square-set stopes. The shaft is vertical.

Table II shows certain statistical data relative to these mines.

Other conditions being equal, stoping costs should be in inverse ratio to the width of the vein, but these ratios will not be proportional. The widest vein has the lowest total stoping cost; and the narrowest vein has the highest cost per ton. The widths between also have this relationship of costs. Evidently, stope preparation and development are almost the same for all these veins; and, if one vein is twice as wide as another, these costs per ton will be only one-half for the wider vein. In drilling and blasting, this ratio is not preserved, because the wider vein will require more holes and more powder; but the number of holes and the amount of powder will be less than twice as much as in the narrower vein. If allowance is made for the higher cost of development work at the Cortez and Mogollon mines, the total cost of mining also has an inverse ratio to the width of the veins. For these veins, 2.90 tons per man-shift for all men underground was produced, and 2.31 lb. powder was required per ton of ore produced, including development work.

Lowest cost per ton, $2.60, was obtained at the Rosiclare mine. The amount has since been raised somewhat

Table I—Cost per Ton of Mining by Shrinkage, Narrow Veins

	Nev.-Mass., Nev., I. C. 6284	Harmony, Idaho, I. C. 6240	Cortez Silver, Nev., I. C. 6327	Mogollon, N. M., A.I.M.E., Vol. 72	Smuggler Union, Colo. *E. & M. J.*, 3/31/28	Rosiclare, Ill., I. C. 6294	Average
Stoping							
Labor	$1.662	$0.650	$1.082	$0.875	.....	$0.700	.....
Supervision	0.064	0.300	0.086	0.154	.....	0.020	.....
Explosives	0.355	0.270	0.304	0.427	.....	0.219	.....
Timber	0.146	0.180	0.054	0.050	.....	0.082	.....
Air, drills, steel	0.100	0.270	0.250	0.186	.....	0.115	.....
Power	0.209	0.160	0.070	.....	.....	0.087	.....
Miscellaneous	0.207	0.220	0.126	.....	.....	0.053	.....
Total stoping	$2.743	$2.050	$1.972	$1.692	$1.654	$1.276	$1.898
Development	0.907	0.650	1.427	1.183	0.396	0.383	.....
Transportation	.....	0.210	0.467	0.880	0.823	0.399	.....
General	0.105	.....	.....	.....	.....	0.404	.....
Miscellaneous	.....	0.230	.....	0.175	.....	0.138	.....
Total mining	$3.755	$3.140	$3.866	$3.930	$2.873	$2.600	$3.721
Width of vein, ft	4.5	5.0	7.0	7.5	10.0	12.0	.....
Year	1928	1929	1929	1922	1927	1929	.....
Daily tonnage	142	150	125	150	700	170	.....

I. C. refers to the U. S. Bureau of Mines Information Circular; A.I.M.E. to *Transactions*, A.I.M.E.; *E. & M. J.* to *Engineering and Mining Journal*.

Table II—Statistical Data, Narrow Veins

	Bd.-Ft. per Ton Stope	Bd.-Ft. per Ton Total	Powder per Ton Stope	Powder per Ton Total	Tons per Man-Shift Stope	Tons per Man-Shift All Underground	Tons per Man-Shift Total
Nevada-Massachusetts	2.90 0.49 lin.	3.63 0.61 lin.	1.65	1.78	.....	2.62	2.50
Harmony	....	....	....	2.05	.....	4.72	....
Cortez	1.36	2.27	2.03	3.30	.....	2.41	....
Mogollon	....	....	1.58	2.62	10.88	1.84	....
Smuggler	....	....	....	....	.....	....	....
Rosiclare	....	1.5 lin.	....	1.80	.....	2.93	....
Average	2.13 0.25 lin.	1.97 0.70	1.75	2.31	10.88	2.90	2.50

Table III—Cost per Ton of Mining by Shrinkage, Wide Veins or Medium-Large Shoots

	Mount Hope, N. J., *E. & M. J.* 7/24/30	Verde Central Ariz., I. C. 6464	Michigan Copper Mine "D," Bureau of Mines Bull. 306	Mine No. 4 Marquette Range, I. C. 6390	Walker, Calif., Annual Rept. 1929	Black Butte, Ore., I. C. 6270	Engels, Calif., I. C. 6260	Average
Stoping								
Labor	$0.339	$0.734	$0.522	$0.418	.....	$0.839	$0.378	.....
Supervision	0.038	0.073	.....	0.036	.....	0.099	0.033	.....
Explosives	0.140	0.234	.....	0.089	.....	0.185	0.157	.....
Timber	.....	0.135	.....	0.011	.....	.....	0.017	.....
Air, drills, steel	0.097	0.485	.....	0.134	.....	.....	0.135	.....
Power	0.123	.....	0.081	.....	.....	0.085	.....	.....
Miscellaneous	0.134	.....	0.237	0.110	.....	0.055	0.040	.....
Total stoping	$0.871	$1.661	$0.840	$0.798	.....	$1.263(c)	$0.760	$1.032
Development	.....	.....	0.361	0.211	.....	.....	0.479	.....
Transportation	0.307	0.580	0.165	0.203	.....	.....	0.438	.....
General	0.067	0.111	0.063	0.053	.....	.....	.....	.....
Miscellaneous	.....	0.033	.....	0.009	.....	0.150	.....	.....
Total mining	$1.245	$2.385	$1.429(a)	$1.274(b)	$1.784	$1.413	$1.677	$1.601
Width of vien, ft.	15.0	15.0	20.0	22.5	35.0	40.0	100.0	.....
Tonnage	238,580	139,203	465,100	131,900	457,637	3,626	303,301	.....
Year	1929	6/1/29 8/1/30	1927	1929	1929	Nov. 1929	1928	.....

(a) These costs are on the basis of tons hoisted, which is 21.65 per cent less than tonnage milled. This amount of waste was discarded in the rock house. (b) Thirty-five per cent of the broken ore left underground. (c) Includes development.

because 25 per cent of the tonnage came from square-set stopes. Perhaps $2.30 would more nearly represent the shrinkage ore cost.

Table III shows the cost per ton of shrinkage mining in seven wide veins or medium-large oreshoots. These mines have been selected from various parts of the United States, so that they may be representative of shrinkage mining.

The following information supplements the details given in Table III: The Mount Hope vein is 15 ft. wide, and dips at an angle of 60 to 70 deg. between gneiss walls, which stand well. The ore, magnetite, is produced at the rate of about 17,000 tons per month. For drilling in stopes, a combination jackhammer with a stoper leg is used; 16-ft. holes are drilled, with a production of 60.63 tons per man-shift in the stopes. The shaft is inclined.

Verde Central orebodies of chalcopyrite and pyrite in a quartz gangue occur as lenses in a larger mineralized zone. The width of lenses varies from 5 to 40 ft. and averages 15 ft. The walls of the stopes usually stand well except in the proximity of faults. In individual stopes the ore may follow one wall of the larger mineralized zone and then change over to the other wall. The country rock is greenstone, intruded by quartz porphyry. The ore is very siliceous and is generally hard.

Michigan Copper Mine "D" is on the Baltic Lode, which is 20 ft. wide and dips at an angle of 70 deg. It occurs in amygdaloid, and the walls are good. Costs are for 1927. The ore in Mine No. 4 is a semi-hard hematite, and occurs as a bed, 10 to 60 ft. wide, in slate. The dip of the ore bed is about 45 deg. In stoping, about 2 ft. of ore is left on the hanging wall, to prevent it from caving.

Walker ore consists of chalcopyrite and pyrite in a gangue of quartz. The vein occurs in black schist and has a maximum width of 120 ft. and an average of 35 ft. It dips at an angle of from 55 to 75 deg. The mine is opened by an adit tunnel 11,000 ft. long and a shaft. Both ore and wall rock stand well.

The Black Butte vein occurs in andesitic rocks and dips at 58 deg. Average stopes are 20 ft. wide, 300 ft. long, and 200 ft. on the dip. The walls are strong. The mine is developed by a tunnel level. Production in November, 1929, was at the rate of 120 tons per day of quicksilver ore.

The Engels main oreshoot has a maximum width of 100 ft. and is 830 ft. long. It occurs in altered diorite, and the walls are firm and stand very well. Stopes are laid off at right angles to the strike of the orebody, with widths of 120 ft. and with pillars 20 ft. between them. Pillars are later recovered by diamond drilling and blasting. The mine is developed by an 8,000-ft. tunnel and an 800-ft. shaft. In 1928, 303,301 tons were produced.

Analysis of the costs in Table III discloses a striking similarity between them. The total stoping cost per ton is nearly the same if allowance is made for certain differences in the ground conditions and the difference in the method of bookkeeping. Increased cost at the Verde Central is obviously due to ore that is harder than that at the other mines. The evidence of this is the fact that the cost for explosives is nearly 10c. per ton higher than the average of the others. The hardness of the ore is also reflected in the air, drill, and drill-steel expense. This item is 36c. per ton higher than the average of the other mines, making a total of 46c. per ton. Hard ore also increases drilling labor. If allowance is made for these items, the Verde Central total stoping cost is closely similar to the average. The cost for the Black Butte includes exploration and development, and with this allowance it, too, is in line with costs of the other mines.

If allowance is made for hard ore, the total mining cost for wide veins is closely similar. Mount Hope has the lowest total cost per ton. In part this is due to the simplicity of the exploration and development work required. On the other hand, this low cost is also the result of the use of the effective method of drilling in stopes, by which 60 tons per man-shift is broken. The character of the orebody at Mine No. 4 tends to make possible low costs.

The conclusion drawn is that given wide veins of medium-hard ore, the total cost of stoping by shrinkage should not greatly exceed $1 per ton; that the total cost of mining should be about $1.50 per ton; and that hard ore may increase these costs by approximately 50 per cent. Table IV gives statistical data.

In regard to large orebodies, a difference of opinion exists as to how the Latouche system of stoping and the method of mining at Alaska Juneau should be classified. The principles of the operation of these methods are identical to those of the normal shrinkage method. The only difference is that the roof of the stope is shot down by a different method.

At Alaska Juneau, instead of drilling horizontal or vertical drill holes in the roof with a burden of from 3 to 6 ft. upon them, coyote drifts are driven in the roof, and these are blasted with a burden of about 35 ft. Stopes range from 200x240 ft. to 250x750 ft. in horizontal cross-section. About 50,000 sq.ft. is considered to be the smallest effective area. Pillars 80 ft. wide are required when the ore is either medium-hard slate or dense gabbro.

At the Beatson mine, by the Latouche system, drilling and blasting are done from raises in the roof of the stope, and

Table IV—Statistical Data, Wide Veins

	Bd.-Ft. per Ton		Powder per Ton		Tons per Man-Shift		
	Stope	Total	Stope	Total	Stope	All Underground	Total
Mount Hope	....	....	....	0.34	60.6	....	.....
Verde Central	....	3.08	....	1.16	....	4.28	.....
Michigan "D"	....	1.35	....	0.95	....	5.20	.....
Mine No. 4 (a)	....	0.01 lin.	0.60	0.90	....	7.58	5.65
Walker	....	....	....	....	....	....	.....
Black Butte	....	....	....	1.16	....	....	4.67
Engels	....	1.23 0.07 lin.	0.81	1.37	27.0	....	5.21
Average	....	1.89 0.03 lin.	0.71	0.98	43.8	5.69	5.18

(a) Thirty-five per cent of ore broken left underground.

Table V—Cost per Ton, Large Orebodies

	Alaska Juneau, Alaska		Alaska Gastineau, Alaska		Beatson, Alaska
	A.I.M.E., Vol. 72	I. C. 6186	A.I.M.E., Vol. 63	A.I.M.E., Vol. 63	A.I.M.E., Vol. 76
Prepaid mining	$0.010	$0.061	$0.050	$0.050	$0.290
Stoping	0.006	0.009	0.284	0.162	0.158
Bulldozing	0.103	0.113	0.084	0.075	0.208
Transportation	0.077	0.114	0.103	0.112	0.137
Miscellaneous	0.017	.....	.....	.....	.....
Stope preparation	.....	.....	0.080	0.080	.....
Totals	$0.213	$0.297	$0.601 (a)	$0.479 (b)	$0.793 (c)
Tonnage	2,310,550	3,670,910	6,523,873	6,523,873	1,682,079
Year	1922	1928	1915–18	1915–18	1923–26

(a) Costs are for shrinkage mining only. (b) Costs are for total production including shrinkage and combined shrinkage and caving. (c) Total costs of Latouche System.

large sections are shot down at one time. Sufficient ore is drawn from the stope to make room for the next slice to be taken across the roof of the stope. The orebody is 800 ft. long and 340 ft. wide. Stopes are 70 ft. wide, laid out across the width of the orebody. Pillars, 30 ft. wide, are mined later by the same method. Entrance to the raises is from above, and men seldom enter the stopes. Ore occurs in slates, graywackes, and schist. The slate and schist ore is soft; the graywackes ore is hard, and breaks in large blocks.

At the Alaska Gastineau, the stopes, when mining schist ore, were typical shrinkage stopes of large size, varying from 40 to 120 ft. wide and up to 400 ft. long. When mining slate ore, stopes were 300 ft. long, parallel to the footwall, and 90 ft. wide, and with pillars 40 ft. wide. The whole area was undercut, and then a narrow slice 7 ft. wide was taken out along the footwall and across the ends of the stope. The ore inside these slices caved. It was necessary to drill and blast only from 15 to 30 per cent of the ore in these stopes, because the remaining ore caved. Table V shows a summary of costs at these mines.

The costs at the mines mentioned in Table V have not been averaged, because the variations of the methods and the resulting costs are so great that the result would be meaningless. Each is remarkable; but, of course, the costs at Alaska Juneau hold the premier position in that they are the lowest underground mining costs in the world.

Estimated Cost of Shrinkage Mining—Costs presented for narrow veins show definite relationships, depending upon the width of the vein. When veins are wider than 15 ft., this relationship of cost to width is not so marked. Wide veins require crosscuts in addition to the longitudinal extraction drift, and additional chutes and raises are also necessary. For this and other reasons a vein 50 ft. wide does not have a much lower cost than a vein 25 ft. wide. In other words, the costs per ton for mining a 50-ft. vein and a 25-ft. vein are not nearly so different as between an 8-ft. and a 4-ft. vein. When a vein is more than about 25 ft. wide, greater width does not necessarily mean lower costs. However, when an orebody attains large dimensions, and enters the present classification of a deposit, length as well as width is essential for low costs. These conditions of size made possible the application of other methods of breaking the ore, which result in lower costs for breaking. Orebodies with large horizontal dimensions, and a large vertical dimension as well, make development and haulage levels unnecessary at every 100-ft. interval. When undercutting is done for an ore column 300 or 400 ft. high, the cost per ton is much less than when the column is only 100 ft. high. In the same manner, the cost for grizzlies and grizzly chambers in the former can be spread over a much larger tonnage. Similarly, other fixed costs can be spread over the large tonnages.

The size of the orebody to be mined by shrinkage is the fundamental condition that determines the cost, provided other conditions are equal. Such a condition does not apply to all methods of mining. Sufficient size and proper shape are essential for the application of caving methods, but caving is different in that the method can be applied only to large orebodies. The size of the individual orebody to be mined by square-set stoping methods has little or no bearing on the cost per ton. With shrinkage and caving methods, within certain limits, the larger the orebody the lower the cost of mining, but this does not apply to square-set mining or to some other methods.

The three classifications of shrinkage stopes already mentioned show costs that range between rather definite limits. The following costs are set down as being typical of each group:

Shrinkage	Costs per Ton: Stoping	Total
Narrow veins		
Width approximately 5 ft.	$1.60	$3.00
Width approximately 10 ft.	1.25	2.50
Wide veins 25 ft. more or less	1.00	1.50
Large deposits	....	0.30–0.80

These are approximate costs only. In the mining of 5-ft. veins, costs might be 50c. per ton higher or lower; with 10-ft. veins, the variation will be 25c. higher or lower. In wide veins the variation will not be more than 25c. either way. For short periods there may be considerable variation, but in the long run they will average up to these figures.

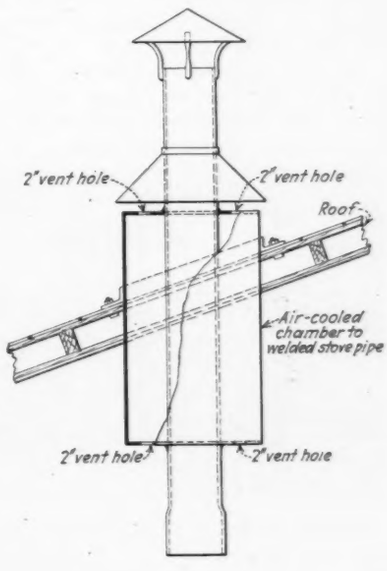
All veins of a given width will not, of course, have the same cost per ton. Other factors besides width determine costs, such as hardness of the ore, continuity of the oreshoots, and whether or not the mine is operated through tunnel or shafts. As already pointed out, hardness of ore may increase costs as much as 50 per cent, and, therefore, the estimated costs presented in the foregoing are for average conditions of ground and development methods.

■ ■ ■

Air-cooled Stovepipe

FAULTILY installed stovepipes constitute a danger to a community where wooden dwellings, staff houses, and offices are maintained. The common plan to overcome this hazard is to fill the void between the stovepipe and the roof with an appropriate insulating material, preferably asbestos—an unsatisfactory procedure, however, as the asbestos ring, generally secured to the pipe by wire bands, may become detached and permit the hot pipe to come in contact with the wooden roof of the building.

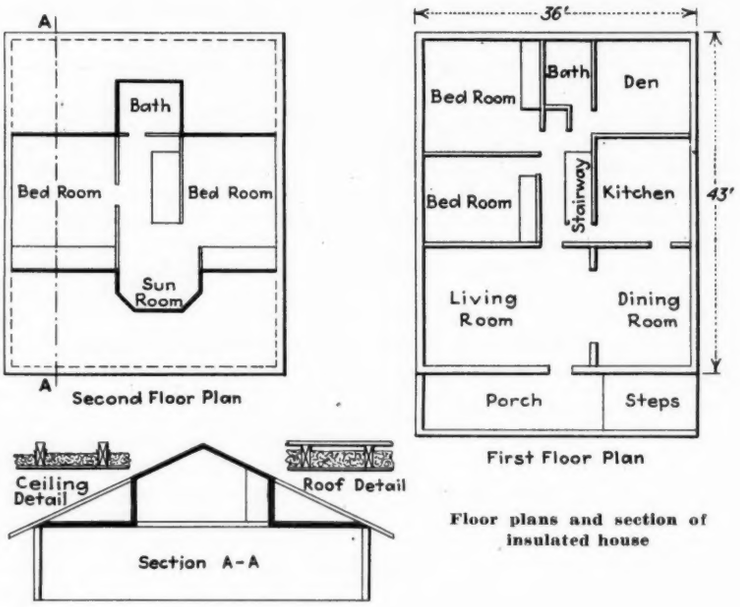
Faced with a similar problem, I. C. Mitchell, master mechanic of the American Zinc Company at Mascot, Tenn., overcame the difficulty by installing the air-cooled stovepipe shown in the accompanying sketch. The pipe contains a welded-on cylindrical chamber, with six 2-in. ventilating holes in the top and six in the bottom. These holes allow air to enter at the bottom of the chamber and leave at the top. A galvanized-iron hood protects the top of the chamber against moisture. The chamber is attached to the roof by a weather-proof circular flange, which is welded to the chamber and bolted to the roof.



Non-Metallic Minerals

For House Insulation

J. B. White



Floor plans and section of insulated house

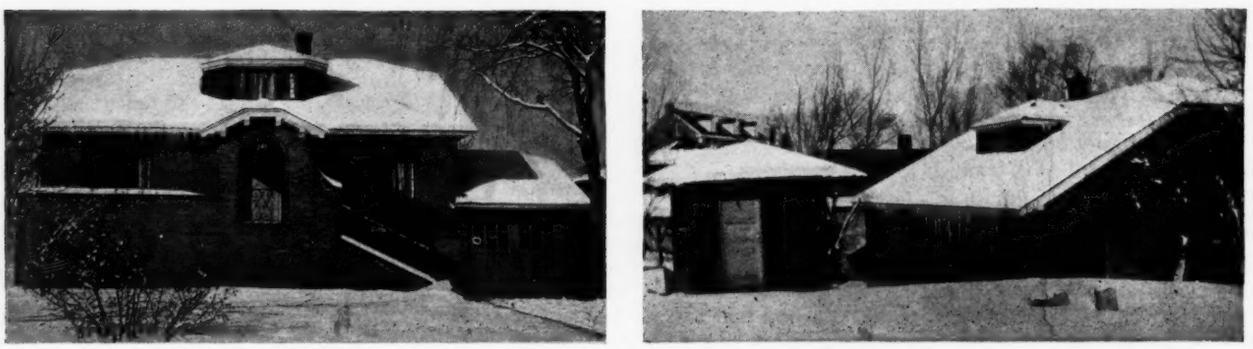
USE of insulation in large industrial structures is not new. Its adaptation for such a purpose has been perfected, and the benefits accruing are apparent, even to the layman. Insulation is usually considered as something used to control the flow of heat only under extremely wide ranges of temperature. The possibilities offered by its use in the construction of a dwelling have been almost completely overlooked.

When insulation is incorporated in the construction of a house the most apparent advantage is one of increased comfort, particularly in a region in which the daily and seasonal range of temperature is wide; the second advantage, if heavy roof insulation is used, is one of additional room space beneath the rafters at small additional cost; the third, if mineral insulation is used, is the satisfaction of having a more fire-resistant dwelling—an advantage further reflected in decreased insurance premiums; and fourth, a smaller furnace installation, with a corresponding saving in initial cost and fuel bills. Each of these advantages will bear enlargement, but perhaps the best summation can be given by referring to a home that was recently built in a Western university town by a well-known mining engineer and educator, according to his own design. The plans, however, were submitted to an architect, who added the lines of architectural beauty.

This house, as shown in the accompanying illustrations, is of one and a half story design. The walls are of brick and the over-all dimensions, exclusive of the porch, are 36x43 ft. The first floor differs little from conventional design, but is notable for the convenience of the arrangement of the rooms and for the space made available in the bedrooms by the use of built-in closets of special design. In the utilization of the upper half-story, particularly, this house differs from the ordinary. Here, in a space usually given up to dead air space or attic—because of the difficulty of controlling temperature in rooms directly under the roof—are two sizable bedrooms that are comfortable in winter and summer, a bath, and a spacious sunroom. In the floor plan of this second story, the heavy lines indicate the insulated walls. The section through this story shows where the insulation was applied, and the details illustrating how the insulating material was poured over the first-floor ceiling and the roof.

The insulating material used on this job is a calcined gypsum and additions which, when mixed with water, evolve CO_2 gas, causing the mass to assume a cellular, pumice-like structure containing a large proportion of small gas cells separated by gypsum partitions. The resultant wet foamy plaster is poured in place and sets within 20 to 30 min. After drying, which takes from two to six days, the material weighs about 12 lb. per cubic foot, and has an insulating value close to that of an equal thickness of cork.

The method of application is not difficult. On the ceilings, where gypsum plaster-board, which had already received its coat of plaster, was used as a lath, the insulation was poured from the tub to a depth of 2 in. directly over black, waterproof building paper laid on the plaster board between the joists.



Snow blanket on house is indicative of its insulated construction. Note snow on adjoining unheated garage and tool house, and absence of snow on roof of a neighboring residence

The Insulex, as it is called, made a tight sealed joint with the ceiling joists. Instead of the usual first move of boarding over the rafters, that portion of the roof directly over the second story rooms was first ceiled with what is known as Celotex, which is a good though thin insulating material itself. The insulating plaster was then poured directly onto the Celotex to a depth of 3 in. No warpage of the Celotex took place, and a tight joint resulted between Insulex and Celotex and rafters. When the roofing boards were laid, the ridge row was not put on for several days, to allow a circulation of air that would thoroughly dry the insulation. In insulating the partitions between the upstairs rooms and the attic space, the Insulex was pre-cast on the floor, in blocks of the required size, and, while still damp, was pushed into place between the studdings and behind the Celotex room lining. The brick walls of the lower floor were not insulated, because the original plan—to have a hollow brick wall, the space between to be filled with Insulex—broke too many precedents for the brick contractor.

The cost of the insulation as placed, considering the advantages secured, was low. A total of 720 sq.ft. of roof and 432 sq.ft. of walls and partition were insulated with 3 lb. to the square foot, and 936 sq.ft. of ceiling with 2 lb. to the square foot. About one-eighth of the total area insulated comprised joists and rafters, and the total amount of insulating material used was 4,700 lb., or about 2½ tons. At $50 per ton, this brought the cost of material to $125.

Fifteen shifts were required to place the insulation, which, at $6 per shift, made a total labor cost of $90. Supplies, consisting primarily of building paper and mixing cans, cost $10, bringing the total cost of this insulation to $225. This cost, however, is not chargeable entirely to insulation when one considers that with its use the only additional expense necessary to add the three comfortable upper rooms was for flooring and fixtures. Moreover, insulation is a relative term: ½ in. of cork or even the same thickness of Celotex board is advertised as excellent house insulation. In the installation considered, a degree of insulation comparable to that of ordinary cold-storage construction is approached by the use of ½ in. of Celotex sheathing, together with 3 in. of Insulex, and this at a modest cost.

The climate of the region in which the house was built offers both daily and seasonal extremes of temperature. During the summer months the thermometer climbs to around 100 deg. on many days; in the winter it hovers close to the zero mark. Under these conditions few houses of ordinary construction can be kept cool in summer, at any cost; and houses can only be kept warm in winter if enough fuel is consumed. A consideration of the question of comfort becomes, therefore, one of comfort alone in summer and of comfort at least cost in winter.

In the insulated home under consideration, the upstairs rooms—constructed without the conventional low horizontal ceiling with dead air space above, but, on, the contrary, with the ceiling following the high rafter lines—are not bake-ovens in the summer, but are as comfortable as the first-story rooms of the average house. Correspondingly, the lower story rooms maintain an unusual and delightfully low temperature.

During the winter a circulating hot-air, coal-fired furnace is used for heating purposes. During the day, temperatures throughout the house are easily maintained at 70 deg. During sleeping hours, halls and bedrooms are open to the night air, but the closed rooms maintain a temperature of 55 to 60 deg., and no night furnace fire is necessary. As a result of this retention of heat in the living room and bathroom throughout the night, the morning furnace firing quickly raises the house temperature to one of comfort. Total coal consumption during the winter of 1930-31 was only 7 tons—a remarkably low figure for a nine-room house in this locality. The ascending heat from the first floor is trapped below the insulated roof, and to use the upstairs hot air registers has been found unnecessary. The insulation is markedly effective as a fuel-saver. This is best evidenced by the retention of snow on the roof for considerable periods of time after a snowfall, until melted by the sun's heat. See the accompanying reproductions of photographs taken two days after a snowfall. These clearly show the roof of the insulated house with its natural insulating blanket of snow still in place, as on the unheated garage; whereas on the roof of a neighboring house, of usual construction, almost all the snow has been melted.

In addition to the use of mineral insulation, this house is an interesting example of the extent to which mineral products may be incorporated in the construction of a modern home. The outside walls are of brick; steel window sashes were used throughout; gypsum plaster board was used to hold the plaster instead of wood lath; the roof is of asbestos cement shingles; the bathrooms are tiled; and even the Celotex used as an interior sheathing of the upstairs rooms is fireproofed with a mineral preparation. The only item that seems to have been overlooked was the possibility of treating the wood used with zinc chloride. This liberal use of mineral products has resulted in a fire-resistant dwelling.

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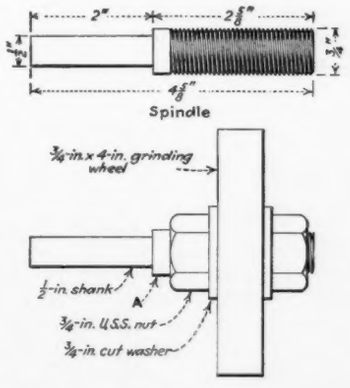
Grinding Pitted Collector Rings

COLLECTOR RINGS on a large alternator, if accidentally allowed to become pitted, offer a real problem to the repair shop. Removing the rotor, so that the rings may be turned or gound, is a lengthy job. Finishing the pitted surfaces with a hand stone is slow, and practically impossible if the pits are deep. Grinding the rings while they are in running position is the best solution. If the shop does not possess a light portable grinder, the electric or air drill may be made to serve the purpose. The only parts of the alternator that must be removed are the brush holders and brushes.



A spindle, of the dimensions shown in the accompanying sketch, is turned in the lathe. The larger diameter of this spindle is threaded within ¼ in. of the shoulder *A*. On this threaded portion is mounted the grinding wheel, which should be of medium-hard grade and of fine enough grain to produce a smooth surface. Its dimensions should be ¾x4 in. The small end of the spindle serves as a shank to be gripped in the chuck jaws of a drill, operated either by electricity or air.

Actual grinding of the collector rings takes little time. If the alternator is driven by steam or water power the throttle valve of the driving machine is opened just enough to turn the alternator slowly. The electric or air drill is then started, and the grinding wheel brought in contact with the pitted surface of the ring. When the required amount of metal has been removed by grinding, a finer finish is given the ring by holding a piece of worn emery cloth against it as it rotates.

If the alternator is driven by a gasoline or diesel engine, the slow speed may be obtained by use of the air-starting arrangement. The engine should not be allowed to start, nor should any attempt be made to do the grinding, while it is rotating rapidly.

Dynamic Braking Of Induction-Motor Mine Hoists

H. W. Chadbourne
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IN LINE with the demand for added refinements in all kinds of control, mine-hoist operators are looking for the ability to use the hoist motor as an electric brake below synchronous speed, thus obtaining better control and saving the mechanical brakes. The only way to accomplish this is by the use of dynamic braking.

Dynamic braking may be used with any hoist equipment where the load is overhauling. The principal application is on unbalanced slopes, especially where the normal speed is high, where speed must be reduced on curves, or where men must be lowered at reduced speed. It may also be used in unbalanced shafts where lowering is desired at less than synchronous speed, or on shafts or slopes, to bring the trip down to low speeds before applying the mechanical brakes to bring the trip to rest. Under any of these conditions the wear on the mechanical brakes is reduced, and the operator has more accurate control of the hoist. Although this article is primarily about mine hoists, dynamic braking can be used also on other types of equipment.

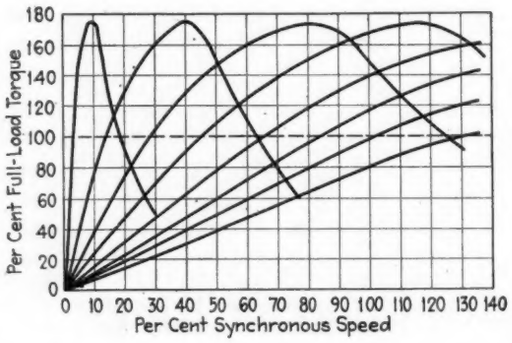

Fig. 1—Dynamic braking curves of slip-ring induction motor

Three kinds of electric braking are in constant use with induction motors: First, regenerative braking where the motor is always connected to the power line, taking a.-c. excitation from the line and being driven as an induction generator, above synchronism. The generator is driven by the overhauling load and the power is pumped back on the line. With this method no control is obtainable below synchronous speed. The second method is plugging or countertorque. When the rotor is in motion, a slip of greater than 100 per cent can be obtained by reversing the direction of the rotating field. The effect of this reversal is to reverse the torque and tend to stop the motor. This system may be used at any speed, but is usually used from normal speed down to standstill; as power is taken from the line and dissipated in the load, it is very uneconomical. The third method is dynamic braking. The first type of dynamic brake was designed and built in South Africa. Instead of using the hoist motor, an independent machine was used, consisting of a water-cooled, cast-iron stator ring and a rotor having a field winding. With the rotor in motion and its field winding excited from a direct-current source, a magnetic flux is set up that cuts the metal of the stator ring. This latter, being a closed circuit, the resulting eddy currents set up an I^2R loss, which gives a braking torque that must be supplied by the overhauling load, and is dissipated in the water-cooling system. The amount of the braking torque is controlled by varying the excitation.

The present system of dynamic braking with an induction motor is based on the fact that a slip-ring induction motor may be operated as an alternating-current generator by opening the a.-c. power line, exciting the stator winding from a direct-current source, and driving it from an external source of power. The speed of the induction generator will depend on the resistance in the secondary circuit, the torque applied, and the amount of direct-current excitation. With a given resistance, any increase in torque applied or decrease in excitation will increase the speed; or, with the torque and excitation constant, an increase in resistance will increase the speed. Fig. 1 shows a set of dynamic braking curves with excitation to give 175 per cent torque maximum. Each curve is a resistance point.

A reduction in excitation gives a similar set of curves, but with a lower maximum torque. Care must be exercised to keep the peak up to a safe value. By changing the resistance and excitation, almost any desired speed may be obtained with a given torque. As will be seen from the curves, any speed may be obtained from 25 or 30 per cent above synchronism down to creeping speed. No reason exists why dynamic braking could not be used at higher speeds, but it rarely is. In fact, at speeds above synchronism, regenerative braking is more often used, thus conserving power. At the low end of

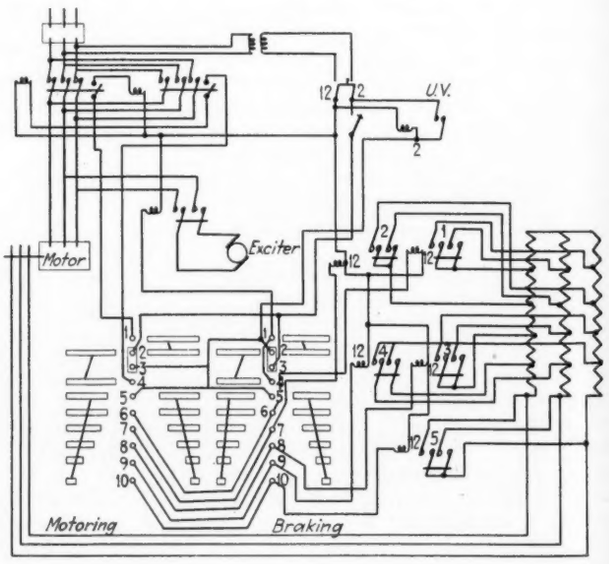

Fig. 2—Convection diagram for dynamic braking

the speed range it is safe to operate down to 15 or 20 per cent of normal. Under certain conditions, creeping speeds can be safely used with magnetic control. Fig. 2 shows a diagram of connections from which the accelerating relays and safety features are omitted.

For operation with dynamic braking, the custom is to use two master switches mechanically interlocked, so that one must be always in the off position. If hoisting is to be done in only one direction, one master switch could be used; but the assumption is that power for hoisting may at some time be required in each direction.

Several methods are available for these master switches, the most common being to use one switch for motoring and the other for braking. Sometimes one switch is used for hoisting in one direction and braking in the other, the second switch being used for hoisting and lowering in the opposite directions.

A difference of opinion exists as to the layout for the dynamic braking switch. One arrangement is for the first point to have all the accelerating contactors open, thus giving maximum resistance in the secondary and maximum speed. Succeeding points lower the resistance and speed. The last point gives creeping speed. If the trip is to be brought to rest, the mechanical brakes are applied, and then the master switch handle must be returned through the higher speed points to the off position. No objection can be raised to this, as the trip is now controlled by the mechanical brakes.

Another plan is to have the first-point braking close all the secondary contactors, thus giving minimum resistance and speed. Succeeding points give higher resistance values and higher speed for any given torque. With this system, when the master switch is used to obtain low speed and then the mechanical brakes are applied, to move the handle from the first point to the off position only is necessary.

For excitation, a direct-current generator is provided, of the proper voltage and current characteristics for exciting the stator winding of the induction-hoist motor. Amount of current depends on the value of braking torque required, or the heating characteristics of the motor. Generally speaking, the excitation voltage is low, varying from 10 to 100 volts, being higher for small motors. The exciter capacity varies from 1 to 3 per cent of the motor capacity. Usually, two phases of the stator winding are connected in series, the third phase being open-circuited, or in parallel with one of the first two phases. The stator winding thus excited becomes the stationary field, and the motor when driven becomes an alternating-current generator, dissipating its power in the secondary resistance. As the full excitation current does not flow through all the windings, the maximum torque, when dynamic braking, is less than when motoring for the same heating.

In addition to standard magnetic control are the extra master switch and exciter set already mentioned. A direct-current primary contactor is also necessary, to connect the exciter circuit to the induction-motor stator winding. As one side of this contactor is connected to the alternating-current line, it must be insulated for full-line volts. Either current-limit or time-limit control for motoring can be used with dynamic braking, but the current-limit is more complicated. An electric tachometer, calibrated in feet per minute, may well be included with each equipment, so that the operator would always know the exact speed of the trip.

Several of these equipments are in use, some having operated successfully for twenty years. However, more new applications have been made in the last five years than before. In 1928, the Tennessee Coal, Iron & Railroad Company installed an 1,800-hp. motor with magnetic control and dynamic braking on a long, unbalanced slope. This was so successful that the necessary new parts were added for dynamic braking to the liquid rheostat control of an older 1,800-hp. unit. Since that time, three other equipments have been installed in that district, one by the T. C. I. & R. R. Co. and two by the Republic Iron & Steel Company.

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Operating Costs at Arizona Smelter Lowered By New Ore-Bedding System

THE new ore-bedding system installed at Copper Queen smelter of the Phelps Dodge Corporation at Douglas, Ariz., in 1930, and described by Mr. David Cole, the designer, in *Engineering and Mining Journal* of Feb. 23, 1931, was operated at decreased cost compared with the old handling methods, and resulted in an improvement in furnace speed and general metallurgical efficiency, according to Mr. Cole. The principal features of the design are a digging elevator for which a trench makes a continuous boot; and the fact that one set of incoming conveyors serves all of the stalls or beds, and one set of outgoing conveyors empties all of them, instead of one set of incoming and one set of outgoing conveyors for each bed, as is the usual practice. The plant has five stalls and 900 ft. of 24-in. belt conveyor instead of the 2,700 ft. that would have had to be installed with the old system.

The following notes regarding the improvement in costs effected by this installation have been courteously supplied by Mr. J. Owen Ambler, smelter superintendent.

The plant was completed in October, 1930. It was designed to handle 68,792 dry tons per month of material, of which 20,900 tons was precipitates and concentrates, the remaining ore and byproducts being crushed in closed circuit to ¼ in. The estimated cost for unloading, crushing, and screening, sampling and assaying, bedding and reclaiming, was 30.1c. per ton, all comparative results being based on February, 1929, wage rates and on 1928 power and commodity prices.

The plant was first started up in October, 1930. Some mechanical troubles with the rake mechanism of the reclaimer were experienced, and operation of the plant as a whole was not normal until February, 1931. By that time the input to the plant had been materially reduced, and for the five-month period February-June, inclusive, averaged only 37,664 tons per month. After June the tonnage received dropped still lower, and the plant was closed down entirely in December, 1931. For comparison with former practice Mr. Ambler has compiled results for the five-month period from February to June, inclusive, 1391.

Equated costs for two periods are shown, being equated on the basis of February, 1929, wage scale and 1928 power and commodities. Former practice is shown by the figures for the first nine months of 1928, from which are excluded all costs of betterment or other unusual expenditures. Present practice is shown by actual equated costs for February-June, 1931.

	Receipts, Month, Tons	Equated Cost Per Dry Ton, Cents
Estimated present practice	68,792	30.1
February-June, 1931	37,664	36.82
Former practice.....	66,685	60.1

Even under the disadvantageous operating conditions prevalent in 1931, an equated saving of 23.28c. per ton was made.

The actual cost for five months in 1931 was 33.7c. per ton, and had the plant operated at designed tonnage rate it would undoubtedly have made or bettered the predicted cost.

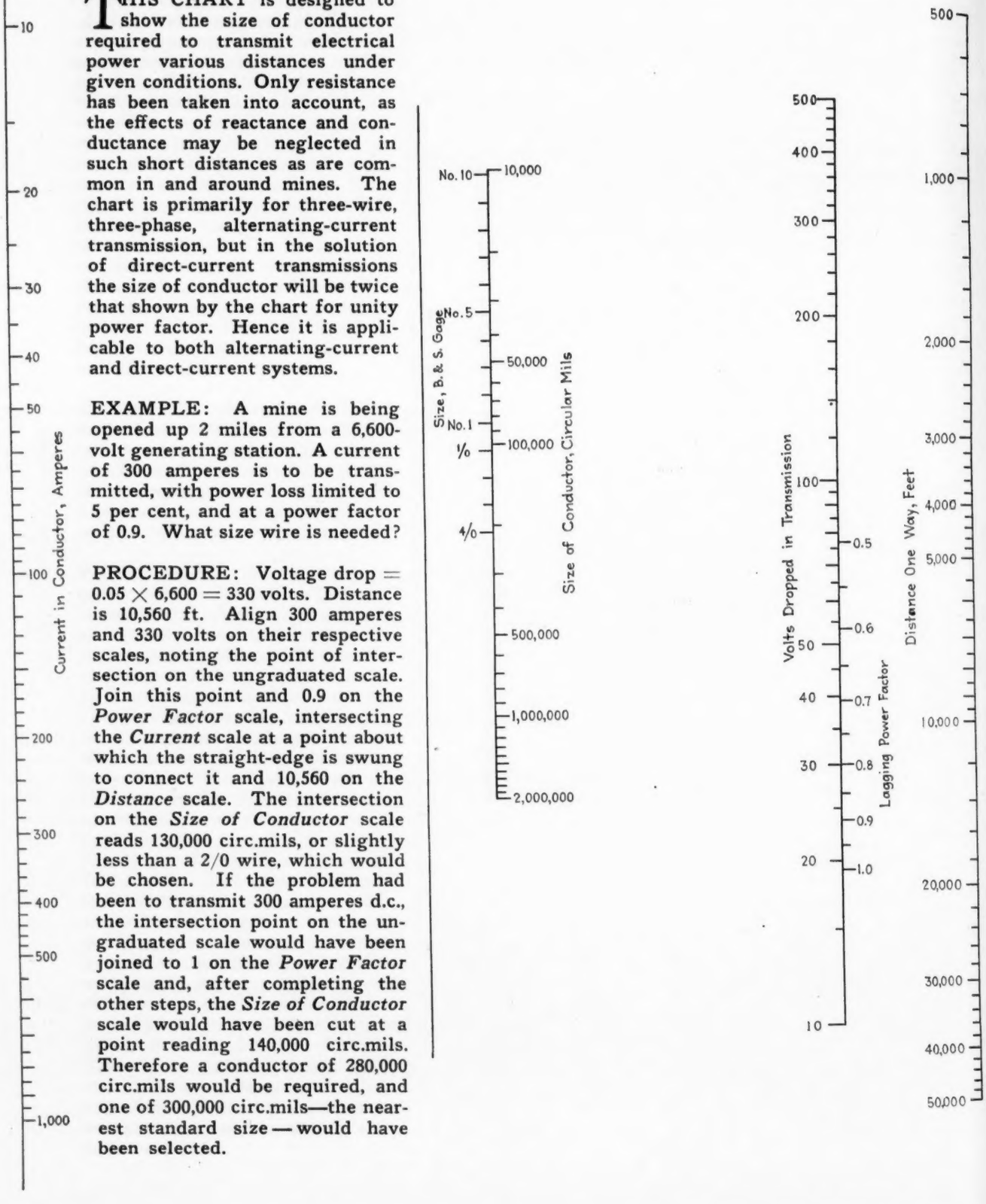
Electrical Transmission Chart

Designed and Copyrighted by
Sydney C. Mifflen

THIS CHART is designed to show the size of conductor required to transmit electrical power various distances under given conditions. Only resistance has been taken into account, as the effects of reactance and conductance may be neglected in such short distances as are common in and around mines. The chart is primarily for three-wire, three-phase, alternating-current transmission, but in the solution of direct-current transmissions the size of conductor will be twice that shown by the chart for unity power factor. Hence it is applicable to both alternating-current and direct-current systems.

EXAMPLE: A mine is being opened up 2 miles from a 6,600-volt generating station. A current of 300 amperes is to be transmitted, with power loss limited to 5 per cent, and at a power factor of 0.9. What size wire is needed?

PROCEDURE: Voltage drop = $0.05 \times 6{,}600 = 330$ volts. Distance is 10,560 ft. Align 300 amperes and 330 volts on their respective scales, noting the point of intersection on the ungraduated scale. Join this point and 0.9 on the *Power Factor* scale, intersecting the *Current* scale at a point about which the straight-edge is swung to connect it and 10,560 on the *Distance* scale. The intersection on the *Size of Conductor* scale reads 130,000 circ.mils, or slightly less than a 2/0 wire, which would be chosen. If the problem had been to transmit 300 amperes d.c., the intersection point on the ungraduated scale would have been joined to 1 on the *Power Factor* scale and, after completing the other steps, the *Size of Conductor* scale would have been cut at a point reading 140,000 circ.mils. Therefore a conductor of 280,000 circ.mils would be required, and one of 300,000 circ.mils—the nearest standard size—would have been selected.



WELFARE AND SAFETY

Policy and practice at the Eagle mine of the Empire Zinc Company, Gilman, Colo.—a New Jersey Zinc Company subsidiary—as described by the staff of the Empire Zinc Company, under the direction of

Russell B. Paul

THE isolation of Gilman, Colo., especially during winter months, requires The Empire Zinc Company to give more than its usual careful consideration to the welfare and safety of its employees at the Eagle mine. At an altitude of nearly 9,000 ft., snow and cold weather are to be expected between Oct. 1 and May 1. The remaining five months consist of one month of spring, three months of summer, and one month of autumn weather. The ordinary maximum temperature is 75 deg. F., and the ordinary minimum temperature is 10 deg. F. below zero. Extreme temperatures are 85 deg. F., as a maximum, and 30 deg. F. below zero, as a minimum. The annual range of temperature in Gilman averages about 85 deg. F. In the summer the mean ranges between 50 and 55 deg. F., and in the winter the mean is below 20 deg. F. Average annual temperature is 33 deg. F.

Accommodations have been provided in Gilman to house about 250 employees comfortably. These facilities consist of about 60 dwellings for married employees and their families, a three-story rooming house, and a two-story hotel, both modern and steam heated. The rooming house and hotel are used by the single employees. Many small cabins are also available. The town is well equipped with a water supply system and a sewerage system. In addition to store buildings, office structure, and the usual industrial buildings, the company owns and operates a hospital and provides clubhouse facilities for the use of the employees. Reasonable rentals are charged for the use of the dwellings; and, likewise, the charge is reasonable to single men for room and board in the rooming house and hotel, which are operated by the Threlkeld Commissary Company. The town of Red Cliff, 3 miles east of Gilman, and the town of Minturn, 6 miles west, also provide accommodations for the employees who prefer to live in those locations.

The U. S. Highway 40-S, also known as the "Pikes Peak-Ocean to Ocean" highway, which extends from Washington, D. C., to Los Angeles, Calif., passes through the town. This highway is maintained the year around. During the winter months additional equipment for removing snow is put into service. In addition to the use of private automobiles for transportation of emyloyees, two buses, operated by private individuals between Gilman and Red Cliff, can handle about 35 men per day. The main line of the D.&R.G.W.R.R., in the Eagle Canyon 600 ft. below the town of Gilman, provides railroad transportation.

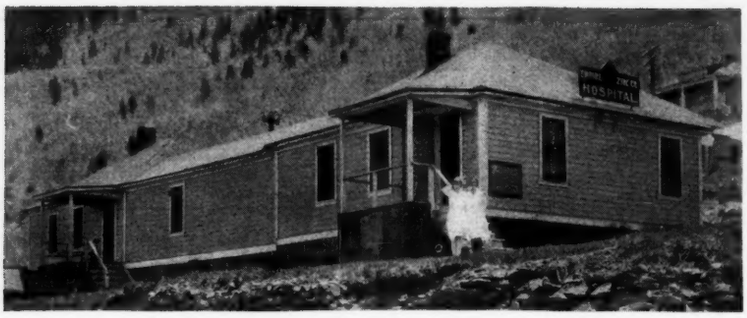
The company hospital, though small, is well equipped. The hospital staff consists of one physician and two registered nurses. Each employee is charged $1 per month, if single, and $1.50 per month, if married or the head of the family. This payment entitles the employee or his immediate family or dependents to medical treatment, when necessary, without further charge. Compensation cases are, of course, handled in accordance with the state laws. The amount collected from the employees as medical dues covers about one-third of the hospital expenses; the remainder is paid by the company. All applicants for employment are given a careful physical examination, and all employees are reexamined once each year.

The Gilman clubhouse, owned and operated by the company, consists of a poolroom, a dance hall, which is also used for motion pictures, and a recreation room for card parties and social gatherings. The recreation room is also used for church services during the winter months.

A complete and modern grade school, with a staff of four teachers including

Gilman, on Battle Mountain, above Eagle River Canyon

This is another of a series of articles on the operations of The Empire Zinc Company at Gilman, Colo.



Company hospital at Gilman for employees

the principal, is maintained at Gilman by the county for children of school age. Present enrollment is 60 pupils. The high school is in Red Cliff, and a bus, owned by the school district, makes daily trips between Gilman and Red Cliff. Gilman has two grocery stores and two small clothing stores, owned by private individuals. The local newspaper, *The Holy Cross Trail,* is published in Red Cliff. It is owned and edited by O. W. Daggett, an active and public-spirited citizen whose experience in mining and milling especially qualifies him for the editorship of a mining district newspaper.

No labor shortage has been experienced for several years. The source of labor is from adjacent mining camps and surrounding farms and lumber camps. Men from the farms work for the winter months, whereas men from the lumber camps work most of the summer months. The general character of all classes of labor is good, and in recent years the number of so-called "floaters" or "ten-day men" has declined noticeably. Nationalities represented among the workers, and the percentage of the total number of workers of each nationality, follow: American, 92 per cent; Swedish, 3 per cent; Mexican, 2 per cent; Austrian, 1 per cent; English, 1 per cent; Italian, 0.5 per cent; Norwegian, 0.5 per cent.

The company is aggressive in maintaining safe conditions both in the camp and underground, and its safety policy is explicitly stated in a pamphlet of safety instructions issued to each employee. This policy is expressed in the following four paragraphs quoted from the foreword of that pamphlet:

The company will use every reasonable precaution for the safety of its employees and will require everyone in its employ to use the utmost care, not only for his own safety but also for the safety of his fellow workmen.

The responsibility for maintaining safe working conditions and for the enforcement of these instructions is placed on superintendents, foremen, and bosses.

Each employee has a definite responsibility for the safe working of his own particular job, and it is expected that he will not only faithfully comply with the safety instructions himself but will also insist that his fellow workmen do likewise.

The violation of any safety instruction will be sufficient grounds for discipline or dismissal.

Safe working is considered a major part of the operating problem and is an essential requirement. No bonuses are paid, nor are contests or entertainments held to promote interest in safety work. The safety instructions are rigidly enforced, and all employees are required to attend classes for instruction in safety work. Subsequently, they are examined relative to their knowledge and understanding of the safety instructions.

During normal times a full-time safety engineer is employed. Departmental safety meetings are held at frequent intervals. The mine meetings are held underground during working hours and are attended by the mine chief, foreman, bosses, and a few workmen. At these meetings the accidents that have occurred and means of prevention are discussed, as well as safety topics of interest. Active assistance from the U. S. Bureau of Mines has been particularly helpful in improving the company's safety work.

Bulletin boards are used to display safety posters and notices. All accidents, whether causing lost time or not, are investigated, and a written report is compiled giving the cause, recommendations for prevention of similar accidents, and a criticism or commendation, as conditions warrant. This report is distributed to the departments interested. The mine accident record at the property during the last three years is presented in the accompanying tables.

Facilities for rendering first-aid treatment are provided both underground and on the surface. Employees are trained to administer first aid, but every man injured is required to report to the company hospital in Gilman before returning to work. Underground workers, except those employed in the underground mill and underground shops, are required to wear safety hats and safety shoes. Safety goggles and belts, when required, are supplied by the company.

Underground fire protection is vigilantly maintained. One watchman is on duty each eight-hour shift, making the round of underground clock-punching stations, one of which is situated in each active section of the mine. Each watchman makes two rounds per shift, so every station is visited at intervals of four hours. Fire-fighting apparatus consists principally of numerous portable fire extinguishers. One or more Pyrene extinguishers are available at each piece of electrical machinery. Soda-acid extinguishers are placed at short intervals in timbered workings and at underground timber-storage yards. Extinguishers are tested monthly. Two fire trucks are kept in readiness for emer-

Summary of Accidents at Eagle Mine, 1929-31

Year	Underground Mining, Number	Surface Mining, Number	Total Mining	Milling, Number	Auxiliary Plants, Number	Grand Total
1929	55	6	61	1	37	99
1930	47	0	47	9	6	62
1931	25	0	25	2	4	31

Lost Time, Man-Hours (*a*)

Year	Total Plant Hours Worked	Lost Time Total Hours	Lost Time Per 10,000 Hours Worked
1929	969,731	9,845	101.52
1930	1,209,571	6,708	55.46
1931	872,438	8,245	94.60

(a) In this tabulation a fatal accident is considered equivalent to loss of time at the rate of 200 hours per month, computed from the date of such accident to the end of the calendar year.

Mine Accident Record, 1929-31

(Mills and auxiliary plants omitted)

Year	Days Mine Operated	Average Number Men Employed	Total Number of Hours Worked	Eight-Hour Shifts Worked	Lost-Time Accidents: Fatal, Number	Lost-Time Accidents: Non-Fatal, Number	Lost-Time Accidents: Non-Fatal, Rate Per 1,000 300-Day Workers	Frequency Per 1,000 Shifts Worked	Frequency Per 1,000,000 Hours Worked
1929	362	190	550,272	68,784	0	61	266	0.887	111
1930	362	265	767,920	95,990	2	47	147	0.512	64
1931	330	223	588,248	73,531	1	24	98	0.340	42.5

Classification of Mine Lost-Time Accidents 1929-31

(Mills and auxiliary plants omitted)

Cause of accident	Per Cent of Total Accidents 1929	Per Cent of Total Accidents 1930	Per Cent of Total Accidents 1931
Falls of rock	30	20	20
Haulage	20	17	24
Falling down chute, winze, stope	11	20	4
All other causes (*a*)	39	43	52

(a) Although the total percentage of "All other causes" is large, the percentage of any one of the included causes is small.

gencies. Each is equipped with two soda-acid extinguishers; one Pyrene extinguisher; 100 ft. of 1-in. fire hose; 16-ft. extension ladders; 18-yd. roll of brattice cloth; hose couplings to fit any underground water line; nails, axe, saw, hack saw, pipe cutter, and wood strips for brattice cloth. The trucks run on the standard track of 24-in. gage that is employed. One truck is kept on stub track off the 14-level haulage lines and the other in the same section on the 16 level.

Six sealed cans, each containing a roll of brattice cloth, nails, and wood strips, are placed at strategic positions underground. Electrical power and light lines are carried through timbered portions of drifts either in conduit or lead-covered cable. Safety switches are used at all electrical installations. Fire protection at the collar of No. 1 shaft consists of two brick fire walls, equipped with self-closing iron doors, that separate the headframe from the change house and other adjacent structures. The shaft and all active portals have a hydrant and fire hose near by. Although carbide lamps are commonly used underground, storage-battery lamps are compulsory for powder men and battery charges. Smoking and open lights are prohibited in the powder magazine and places where combustibles are stowed. Smoking is not prohibited generally underground, but the company has requested employees not to smoke in timbered stopes, headings, or around timber of any description. The response to this request has been gratifying.

Owing to Gilman's situation on the side of Battle Mountain, severe winters, extensive underground workings, and many stopes in heavy ground, operating hazards both on the surface and underground are more serious than those that exist in many mines and mining camps. However, with careful supervison and the cooperation of employees, the company has succeeded in continually improving its accident records. A determined effort exists not only to continue to reduce lost-time accidents but also to reduce greatly the trivial or no-lost-time accidents, which are too numerous and, in the aggregate, expensive.

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Shaft-Sinking Record at Johannesburg

Owen Letcher

WHAT is believed to constitute a record for this type of shaft was accomplished at the West Rand Consolidated Mines, Johannesburg, Transvaal, during March, 1932, when 400 ft. was sunk. The shaft is of five compartments, each 10x5 ft., outside measurement of timber being 29 ft. 6 in. by 11 ft. 6 in. The depth of the shaft on April 1, 1932, was 1,113 ft. from the surface. It will probably be stopped at a depth of about 4,000 ft.

Sinking commenced in 1929, when 86 ft. was sunk through the Black Reef formation. This distance was heavily concreted, and surface bearers were put in prior to the erection of steel headgear. The shaft was timbered to within 15 ft. of the bottom and allowed to fill with water. On Dec. 31, 1931, sinking was restarted, and 7 ft. was completed. The footages sunk monthly thereafter have been as follows: January, 303 ft.; February, 317 ft.; March, 400 ft., thus making the total depth on April 1 of 1,113 ft.

Timbers used are 9x9 in. wallplates, end plates, and dividers; 8x8 in. corner posts; 10x4 in. studdles, with set centers of 7 ft. 6 in. The guides (8x4 in.) are 30 ft. long. Nos. 4 and 5 compartments are fitted with Karri guides. Nos. 1, 2, and 3 compartments are fitted with pitch-pine guides. The shaft is served with three winding engines—No. 1 compartment with a small engine used in connection with timbering and pipe installation; Nos. 2 and 3 compartments with a Petzold hoist, operating skips with an effective capacity of 2.25 tons; Nos. 4 and 5 compartments with skips of the same size as for Nos. 2 and 3 compartments, the engine being a drop-valve Robey hoist. The bearer sets in No. 1 compartment are at intervals of 100 ft.; in the other compartments they are at intervals of 200 ft.

Cost of sinking for March, including timbering, was £24.13.8 per foot. The European staff is 28, and the native staff 250, on three 8-hr. shifts; 91 rounds were blasted during the month. The machines used are Chicago Pneumatic No. 5 drills, converted to sinking machines; an average of ten machines was used per shift.

No special method was adopted for lashing. A low-built skip facilitated high loading speed. Broken rock was hoisted by the two engines, and the usual time of cleaning out a round was about 4½ hr., from starting until the shaft bottom was ready for drilling. On an average, 140 tons of rock was removed from the shaft bottom in this time.

The type of explosives used varied in accordance with the nature of the ground. In March, about the same quantity of 60 per cent Gelignite was used as of 50 per cent, the size of cartridge being 1x8 in. Quantity per blast ranged from 130 to 150 lb.

The shaft is ventilated from the surface with a high-speed Sirocco fan, blowing air directly to bottom through 22-in. galvanized-iron piping. The drills are served with a 6-in. air column; and a 2-in. column supplies water. The steel used is 1 in., hexagonal. Number of bits per hole ranges from two to five, according to the nature of the ground.

The shaft, when completed, will serve the development of the Main Reef Series below Level 27, which areas are served in the Main Reef Section by the East and West Incline shafts. The shaft will probably intersect the Main Reef at about Level 34, or a vertical depth of about 3,500 ft. below the surface, and will connect with the present workings at the plane of Level 32. It will be equipped with a pumping plant sufficient to handle all underground water from both the Main Reef and Battery Reef series. When completed, in addition to serving the Main Reef series, it will permit hoisting all labor from the Battery Reef workings. When the ore in the Main Reef series has been worked to the maximum depth of the present used Main Reef shafts, the South Shaft will be called upon to supply the major part of the ore for the reduction plant.

On April 1, at a ceremony at the shaft head, Sir George Albu, chairman of West Rand Consolidated, presented T. R. Boyde, the master sinker, with an engraved silver salver in recognition of the achievement of himself and his men. Tribute was also paid to C. S. McLean, general manager; W. H. Claydon, consulting mechanical and electrical engineer; Errol Hay, technical adviser; the foreman, timberman, and the native workers.

STARCH In The Flotation of GOLD ORES

D. N. Vedensky
and
L. H. Duschak

Soluble starch can be used effectively as a depressant of certain minerals, the results being especially advantageous where the ratio of concentration is high

IN RECENT YEARS organic substances such as starch and various gums have been used in flotation, usually to influence the behavior of some constituent of the gangue. A patent by R. S. Phelan and S. P. Lowe (U. S. Patent No. 1,690,907, June 6, 1928) describes the use of starch to prevent the flotation of micaceous and talcky minerals in floating sulphides. The illustration given in the patent specifications deals with the recovery of a copper concentrate from a sulphide ore containing pyrite. By the use of starch in an acid circuit, mica is depressed and the grade of copper concentrate materially increased. Fahrenwald (*Trans.* A.I.M.E. Vol. 70, p. 688, 1924) also mentions starch in discussing the action of flotation reagents.

Recent experience has shown that starch can perform a particularly important function in the flotation of certain gold ores. The quantity of valuable minerals in these ores, usually auriferous sulphides and fine, free gold, is often very small. With such ores the theoretical ratio of concentration is high and the possible grade of concentrate correspondingly high. Evidently, when the quantity of gold-bearing minerals is only 1 to 2 per cent of the weight of the ore, the contamination of the concentrate with a few per cent of gangue minerals will greatly reduce the grade of concentrate, but not to the same extent as when floating base-metal sulphides where the weight of concentrate bears a much larger ratio to the weight of the original ore.

The writers of this article collaborated with Max Kraut, of the Pan-American Engineering Corporation, in working out the application of starch in floating amalgamation tailing at the two mills of the Idaho-Maryland Mines Consolidated, at Grass Valley, Calif. These mills have the stamp-milling equipment characteristic of the district. The tests, summarized in this article, were made on amalgamation tailing coming from the plates when 40-mesh screens were used in the mill. Both tests were made in Kraut laboratory flotation machines. In Test A no starch was used. In Test B starch was used at the rate of 1 lb. per ton of ore in the rougher and 0.2 lb. per ton in the cleaner.

The pulp coming through a 40-mesh screen was obviously too coarse for the best flotation results. On regrinding, a tailing was obtained from flotation carrying only 0.018 oz. gold.

Tests Made by Idaho-Maryland to Determine Effect of Starch

Test "A"—Using No Starch

Product	Weight, Per Cent	Au, Oz.	Au, Per Cent
Heads	100.0	0.16	100.0
Concentrate	13.22	0.98	80.9
Tailing	86.78	0.035	19.1

Test "B"—Using Starch

Product	Weight, Per Cent	Au, Oz.	Au, Per Cent
Heads	100.0	0.16	100.0
Concentrate	2.11	6.23	81.7
Tailing	97.89	0.03	18.3

In both of these tests, the pulp was alkaline, having a *pH* value of approximately 7.6. This alkalinity was due to the water and gangue constituents of the ore, no alkali being introduced with the starch. Thus the favorable action of starch in depressing gangue minerals has been obtained in alkaline pulp as distinguished from the application described by Phelan and Lowe, in which an acid circuit is specified.

The ore from the Idaho-Maryland mine contains a large amount of talc and serpentine, which have a strong tendency to float. In fact, when a flotation test is made without the use of starch, these are the first minerals appearing in the froth.

Starch was found to exert a depressing effect not only on the talc and serpentine but also on the gold-bearing minerals, which include pyrite, galena, gold telluride, and possibly fine free gold and minute particles of amalgam. Subsequent tests showed that the depressing effect on the gold-bearing minerals could be overcome by the use of a suitable activator such as copper sulphate, and that by properly proportioning the starch and copper sulphate, the gold-bearing minerals could be floated and the talc and serpentine largely depressed. Many attempts were made to obtain satisfactory flotation without the use of starch, but in all of them the quantity of talc and serpentine which floated reduced the grade of the concentrate below economic limits.

In regular milling practice, two major factors called for consideration in working out the application of starch. Almost at the start the fact was recognized that ordinary starch was entirely unsuitable and that the desired effect could be obtained only by using so-called soluble starch. The problem with respect to this factor was to work out a practical and inexpensive method of making soluble starch which would produce the desired effect. This subject will be discussed hereafter in this article. The second factor was the determination of the other flotation conditions under which the best results from starch could be obtained. As just mentioned, starch in all forms, including the soluble kind, exerts a depressing action on the gold-bearing minerals. A method of offsetting this by use of a suitable activator had therefore to be devised. Small-scale tests showed that, to obtain optimum results, a rather delicate balancing of conditions was necessary and that the final work on this would have to be done in the mill rather than in the laboratory.

The first attempt to use starch on a large scale was made at the Brunswick mill of the Idaho-Maryland company, under the immediate direction of Ernest W. Ellis, mill superintendent. By following a systematic program of testing that involved a large amount of detailed and painstaking work, Mr. Ellis established the optimum adjustment of flotation conditions. As soon as this had been done, the mill results fulfilled the expectation based on the small-scale tests. Following this successful work at the Brunswick mill, the use of starch was introduced at the Idaho-Maryland mill, where the problem is substantially the same.

In the early work, the starch used was the so-called "soluble starch" commonly sold as a laboratory reagent. This is expensive (35c. per pound), and inasmuch as approximately 0.6 lb. was required, per ton of ore floated, cost was an important item. Various less expensive grades of starch de-

scribed as "soluble starch" were tried, but they were found unsatisfactory. Moreover, laboratory tests indicated that common starch is a strong depressant for gold and gold-bearing minerals, producing in flotation just the reverse effect of that desired. This experience led to a study of the methods of preparing soluble starch, which was undertaken by Mr. Kraut and the writers of this article. Evidently, unless the cost of soluble starch could be greatly reduced, its use on a commercial scale in flotation would be at least greatly restricted.

Common starch is insoluble in cold water. In hot water it swells, forming the familiar starch paste, but does not pass into true solution. The paste so obtained becomes viscous when cold and is very difficult to handle in any reagent feeder. Prolonged boiling of a dilute starch paste renders part of the starch truly soluble, but the unconverted remainder is detrimental, as mentioned in the foregoing. A better yield is obtained by cooking the starch under pressure at 150 deg. C.

Several other methods of preparation were investigated, including treatment with mineral acids. The procedure found most satisfactory consisted in boiling starch paste in a dilute solution of sodium hydroxide. With a 2 per cent solution of sodium hydroxide, a solution containing 5 per cent of starch can be prepared. This treatment renders the starch completely soluble and leaves no unconverted residue. Only approximately 20 per cent of the sodium hydroxide is actually consumed, and possibly by longer boiling a lower concentration of the alkali would produce the desired result. The cost of soluble starch prepared in this way is estimated to be 8c. per pound. The original source of the crude starch appears to be unimportant, as preparations made from potato starch and from cornstarch gave equally good results.

Stability of the solution of soluble starch thus produced by caustic treatment is increased if the excess alkali is neutralized with mineral acid to the methyl-orange end point. Moreover, caustic alkalinity is frequently undesirable in gold flotation because of its depressing action on gold and gold-bearing minerals.

The method of preparing soluble starch just described was introduced at the mills of Idaho-Maryland Consolidated by Mr. Kraut. Current practice is to prepare a 4 per cent starch solution, using a 1 per cent solution of sodium hydroxide. Excess alkalinity is neutralized with sulphuric acid before feeding the starch to the flotation circuit. The pulp is distinctly alkaline, however, owing to the slight solubility of various constituents of the gangue. This natural alkalinity appears to be in no way detrimental, whereas introduction of additional alkali with the starch affects results adversely. A total of 0.6 lb. of starch per ton of ore is used in the rougher and cleaner units. The other reagents employed are American Cyanamid No. 301, copper sulphate, and cresylic acid. Copper sulphate is used as an activator to offset the slight depressing effect of the soluble starch on the sulphides.

Following the successful application of soluble starch prepared by the sodium hydroxide method in the Idaho-Maryland operations, it was tried experimentally in another California mill where stamp-mill tailing is being floated. Here the purpose was to depress amorphous carbon, which tended to dilute the gold concentrate. A small amount of starch added to the cleaner practically doubled the grade of concentrate without adverse effect on the tailing.

The experience of Mr. Kraut and the authors' indicates that soluble starch may be useful in many flotation operations. The fact must be recognized that various commercial brands of starch offered as "soluble starch" may be entirely unsatisfactory for flotation use. To obtain soluble starch in solid form, it must be separated from the dilute solution in which it is orginally prepared. This may be done either by adding alcohol or evaporating the solution. Such methods are expensive, which no doubt accounts for the high cost of true soluble starch in solid form. For flotation work, preparation at the point of use by the method described will be more economical.

Soluble starch has been used experimentally in a number of laboratory flotation tests involving the depression of one or more constituents. These tests, although not conclusive with respect to any specific application, suggest that properly prepared soluble starch may find various uses in flotation practice. For example, discovery was made that in separating cinnabar from an ore containing pyrite and an amorphous carbon, soluble starch materially increased the depressing action of lime on the pyrite and also depressed the amorphous carbon. In general, soluble starch may be said to exert a depressing effect upon both metallic and non-metallic constituents of an ore pulp. This fact may be used to advantage, in making flotation separations, by employing an activator which is specific for the mineral to be floated.

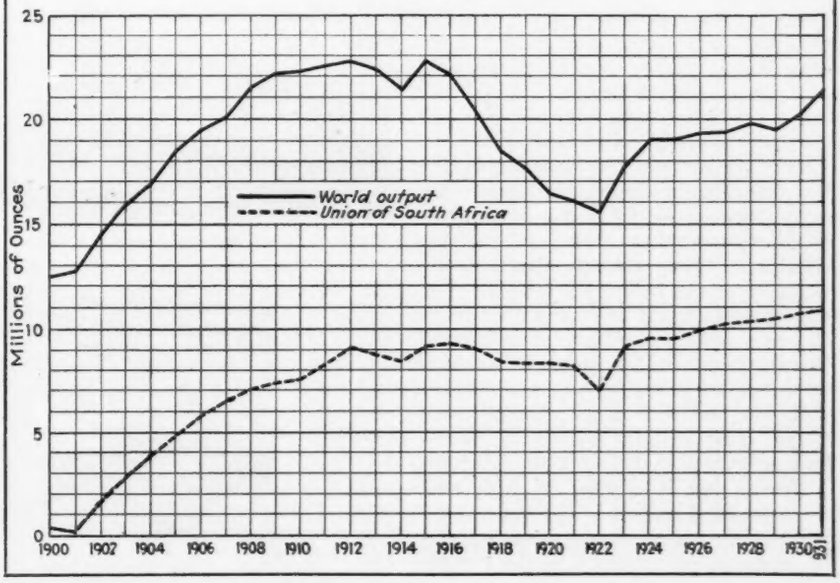
A few tests using dextrin in lieu of soluble starch were made. These indicate that the action of dextrin and soluble starch are generally similar. Possible use of these reagents in flotation work appears worthy of investigation.

The writers of this article are glad to acknowledge their indebtedness to Mr. Kraut for suggestions in the course of the work, and to Ernest W. Ellis, superintendent of the Brunswick mill, for information concerning the results actually obtained with soluble starch in regular milling practice.

The results of our experiments may be summarized as follows:

1. Soluble starch has been found useful in connection with the flotation of gold ores in California. Under properly controlled conditions, it exerts an important depressing effect on certain readily floatable constituents of the gangue.
2. The useful effects can be obtained only by employing true soluble starch.
3. An inexpensive method of preparing soluble starch at the point of use has been worked out.
4. True soluble starch may possibly have other applications in flotation.

World gold production from the great Witwatersrand mines of South Africa and elsewhere has steadily increased in recent years. South African output set a new record in 1931. World output may set a new one in 1932.



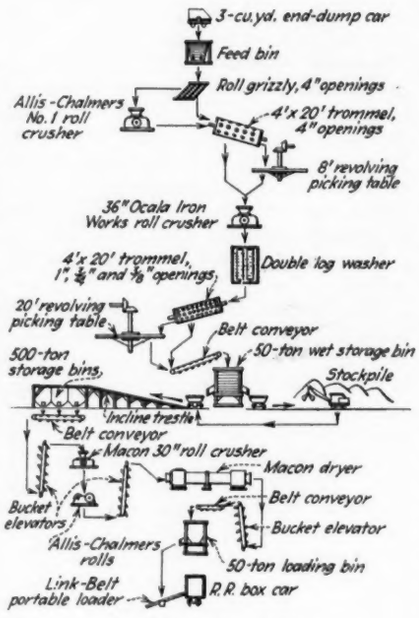
Mining and Processing Hard-Rock Phosphate

J. B. Huttl

FERNANDO COUNTY, FLA., noted for its hard-rock phosphate deposits, has been an important producer of phosphate for years. The deposits, unlike those in the land-pebble belt to the south, are irregularly scattered. They occur in shallow beds, from 5 to 50 ft. thick, as fragmentary rock, boulders, plate rock, or pebbles of Pleistocene age. Rock of commercial value rarely exceeds 25 per cent of the deposit. Overburden, consisting chiefly of sand and clay, averages 20 ft. The individual phosphate fragments, as they occur in the clayey deposit, range from grains to boulders weighing more than 100 lb. Color of the exterior varies from a yellowish gray to brown.

The Felicia mine, of Camp Phosphate Company, about 7 miles north of Inverness, Fla., is at present the major operation engaged in exploiting the hard-rock deposits in this district. A ½-mile spur, connecting with the main line of the Atlantic Coast R.R. at Felicia, affords access to the mine and the washery. The mine consists of an irregular-shaped pit that varies in depth from 75 to 100 ft., depending principally on the contour of the surface. Stripping of the 25-ft. overburden is done by hydraulic methods. Two 6-in. monitors, fed from a 10-in. main, are available for this work. Water is obtained from a 12-in. well, whence centrifugal pumps deliver it at 200-lb. pressure to the portable hydraulic guns. All phosphate-bearing matrix below the permanent ground-water level of the region is mined by a coal-fired Tampa type dredge, equipped with a 40-ft. boom and a 1½-cu.yd. bucket; that above the ground-water level is removed by a ¾-cu.yd. Bucyrus-Erie steam shovel mounted on a caterpillar truck. Construction of the dredge resembles that of a steam shovel mounted on a steel hull. Broken rock is loaded into specially designed end-dump cars of 3-cu.yd. capacity. The cars are hauled by a four-drum friction hoist and cable over a wooden incline and a 36-in.-gage track system that rests on wooden trestles to the crushing floor of the washery at the mouth of the pit. Pile pulling and driving is done with a small steam-driven pile driver when a new location is desired. Large rock boulders, occasionally encountered, are drilled with IR Jackhamers, and blasted with 40 per cent gelatine dynamite. Total tonnage mined a day exceeds 300 tons.

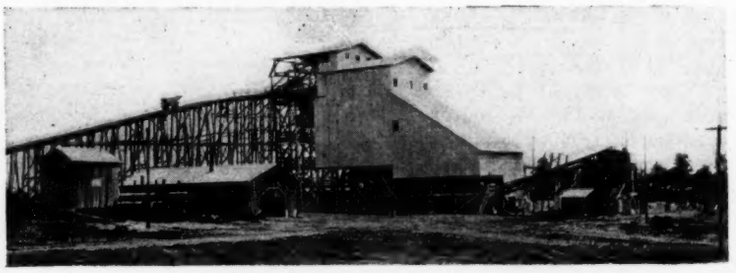
Prospecting is carried on with well drills or hand augers. The auger used is a modified form of post-hole auger, 4 in. in diameter. Because of the irregularity of the phosphate deposits, more drill holes are required to test a new tract of land than are necessary in the land-pebble area. A record is kept of each hole drilled, showing thickness of the overburden and of the phosphate-bearing matrix, as well as sampling results. These data are used to calculate the extent and value of the tract.



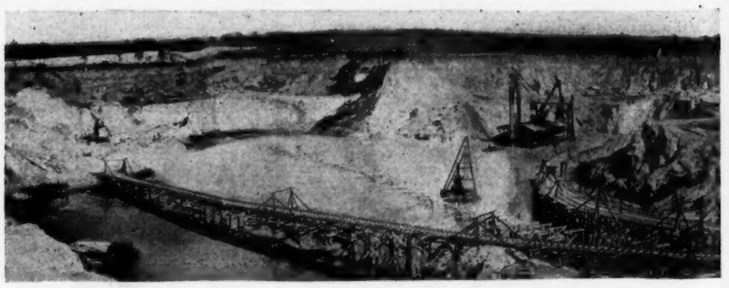
Flowsheet of washery

Processing of the mined phosphate rock is relatively simple, as indicated in the accompanying flowsheet. At the washer the rock is dumped into a steel-lined feed bin fitted with a steel roll grizzly, with 4-in. openings. Undersize goes to a 20-ft. revolving trommel, with 4-in. openings, and the oversize to an Allis-Chalmers No. 1 roll crusher. Material discharged at the end of the trommel falls onto an 8-ft. slowly revolving picking table, from which waste is removed manually. The phosphate rock on the table, together with the undersize from the trommel, passes to a 36-in. Ocala Iron Works roll crusher, the product flowing by gravity to a double-log washer. Water is added at the washer, the clayey overflow of which passes to the mud pond. The washed rock goes to a 20-ft. trommel equipped with screens. Oversize of the trommel is hand-picked on a 20-ft. picking table, the picked product, together with the trommel undersize, passing by belt conveyor to a 50-ton wet storage bin. Small mine cars either distribute the contents of the bin to the stockpiles for rewashing and drying, or move it over an incline trestle to the 500-ton storage bins of the adjacent drying plant. A small caterpillar-mounted P&H gas shovel, with ½-cu.yd. clamshell bucket, is available for reloading cars at the stockpiles.

At the drying plant the phosphate rock is dried and crushed to specifications. From the storage bins at the plant the rock passes successively through a 30-in. Macon roll crusher, a set of Allis-Chalmers rolls spaced to produce a ⅜-in. product, and a 40-ft. oil-fired Macon dryer. Ultimately, the dried material is elevated and conveyed by a short belt conveyor to a 50-ton loading bin, from which it is loaded into box cars by means of a portable, gasoline-driven, Link-Belt loader.



Washery treating output of the Felicia mine



Pit of the Camp Phosphate Company near Inverness, Fla.

Electrical Prospecting in Canada

J. J. Breusse

Société de Prospection Electrique, Paris, and Schlumberger Electrical Prospecting Methods, New York and Toronto

THE purpose of this paper is to illustrate, by two examples of practical surveys, the use of the self-potential method in ore exploration. This method was discovered in 1912 by Prof. Conrad Schlumberger, and was in practice in Europe before the War. It was introduced into the western hemisphere, and to Canada in particular, in 1921, by Sherwin F. Kelly, being instrumental in the discovery of the *E* and *G* orebodies at Noranda Mines, Quebec, in 1924.

For the reconnaissance of mining claims, recourse may be had to various geophysical processes. Even considering only the electrical methods, the geophysicist may choose the self-potential method; the potential method (by the study of the deformation of equipotential curves, or by resistivity measurements); and the electro-magnetic method. Each has its advantages and limitations, and, according to the problem under consideration, one of them may be especially recommended. Mathematical and physical principles underlying these methods have been outlined in technical publications,[1] to which the reader is referred for detailed information. This article will discuss briefly the conditions under which the phenomenon of spontaneous polarization occurs, and the reasons why it is one of the most reliable and rapid techniques in the exploration for ore.

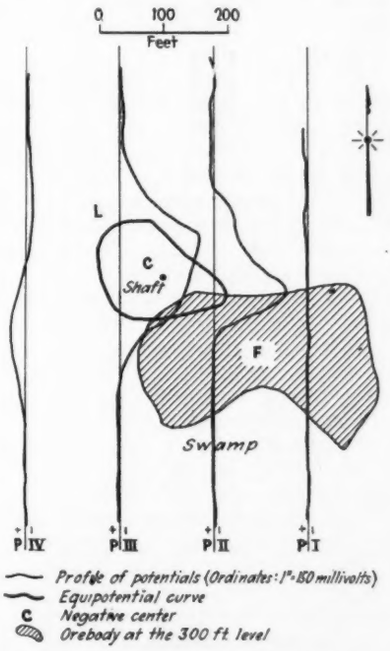


Fig. 1—Results of self-potential survey over F orebody, Amulet Mines, Quebec

A body of metallically conductive sulphides buried in the ground can be compared to a conductive mass plunged in a non-homogeneous electrolyte. At the upper part of the body, near the surface, the water contained in the rocks is rich in oxygen; at depth the reverse condition occurs. An electrical battery is then constituted by the conductive ore and the surrounding rocks. It will act in such a manner as to correct the dissymmetry existing between the upper part and the lower part of the orebody. Negative ions will be carried toward the top of the orebody, where oxygen exists in abundance. As a result, a negative region of potentials will be observed at the surface, on the apex of the orebody. Such a zone, toward which the current is flowing, is called a negative center.

The object of the self-potential survey of a region is to locate the negative centers existing within it, the conductive deposits being regularly connected with them. This indicates one of the advantages of the self-potential method—that an indication discovered by it corresponds to a mass possessing a metallic conductivity. Such an indication cannot, in any instance, result from a mere difference in the conductivities of the rocks, this latter phenomenon being of an electrolytic nature. With several other electrical methods of exploration experience, difficulties are encountered in interpreting field data, on account of numerous pseudo-indications of conductors—such as wet faults, geological contacts, thin conductive beds in a resistant formation — caused by variations in conductivity of the underground rocks.

Some authors have stated that the phenomenon of spontaneous polarization may not exist for certain types of orebodies. This is correct if the deposit is not metallically conductive, such as is the condition with a body of galena or pyrite crystals inclosed in a quartz or calcite gangue. But, in such case, it must be borne in mind that no other method of electrical exploration will detect them. My experience, on the contrary, proves that the self-potential method always detects the metallically conductive orebodies. This is in accord with the opinions of numerous geophysicists who consider the process as the most efficient that can be utilized for ore exploration.[2]

Under one circumstance, however—with graphitic schists—the self-potential methods will register numerous indications of spontaneous activity that do not correspond to buried mineralization. This is natural. Graphite is a metallically conductive mineral. In this application of the method, the importance of the reactions, their size and nature, will generally permit the geophysicist to differentiate them from the indications caused by metallic deposits. If necessary, such a differentiation will be completed by a summary geological reconnaissance.

The remarks presented in the foregoing do not infer that other methods of electrical exploration are not efficient or interesting. On the contrary, every process has its special advantages and field of usefulness. In particular, in the course of an electrical survey, the results obtained by one method should be controlled and supplemented by checking against those from a second independent process. In this connection, the potential and electromagnetic methods will prove valuable in the detailed study of the indications discovered in the course of a self-potential reconnaissance. These technical qualities constitute only a part of the advantages of the self-potential method. To them must be added the fact that

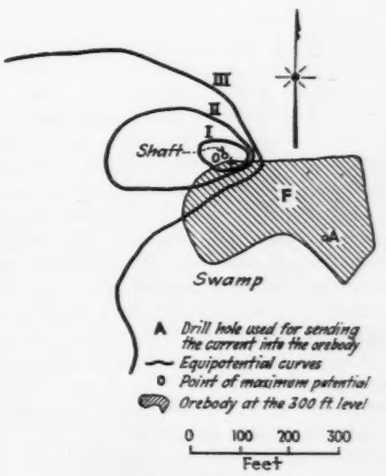


Fig. 2—Equipotential curves obtained by sending an electric current into F orebody by means of insulated wire lowered into drill hole *A*

[1] E. G. Leonardon and S. F. Kelly: Canadian Inst. Min. & Met., Bulletin, January, 1928. Max Mason: Technical Publication 45, A.I.M.E., February, 1928. Etienne S. Bieler: Can. Inst. Min. & Met., Bulletin, May, 1928. E. H. Guilford: Can. Inst. Min. & Met., Bulletin, May, 1928. Hans Lundberg: Can. Inst. Min. & Met., Bulletin, May, 1928. E. G. Leonardon and S. F. Kelly: Can. Inst. Min. & Met., Bulletin, May, 1928.

[2] Cf. Canadian Geological Survey. Memoir 165. Studies of Geophysical Methods, 1928 and 1929.

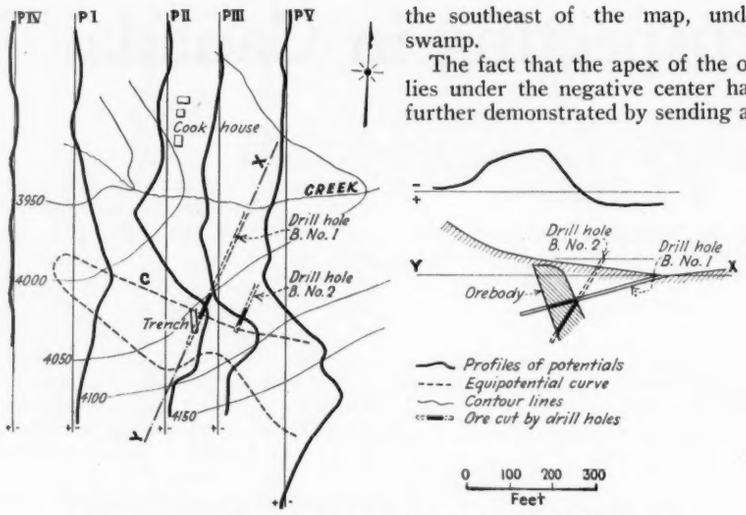


Fig. 3—Surface map and cross-section illustrating the discovery of an orebody at Hope, B. C., by self-potential measurements

the apparatus is easily transportable and that the measurements can be made speedily. In consequence, the cost of exploration per acre is reasonable; it is the cheapest and most reliable method of exploration for ore in virgin territory.

An electrical survey of the *F* orebody of the Amulet Mines, in the Rouyn district of Quebec, was of particular interest. The area had been previously examined by various geophysical operators, without satisfactory result. Geological and physical conditions of the ore occurrence are complex. The orebodies are found at the contact of two lava flows (rhyolite overlain by a flow of andesite), on the eastern flank of a broad anticline, the direction of which is N.N.W.—S.S.E. In the central part of the anticline, the andesite has been eroded, and rhyolite lavas outcrop. Toward the east, these rocks dip gently under the andesite flows. Of significance in electrical prospecting is the fact that contact between the two eruptive rocks is well mineralized on most of its surface with a thin sheet of conductive sulphides. Part of the uninterpretable results obtained with the electro-magnetic methods were probably a consequence of the presence of this thin and continuous layer of conductive material.

Fig. 1 illustrates the results obtained by the self-potential survey over the *F* orebody. Four profiles of potential were made. Two of them encountered strong negative reactions caused by the upper part of the deposit. The detailed work consisted in determining accurately the position of the negative centre *C*, and tracing an equipotential curve *L* around it.

On the map, the horizontal section of the commercial orebody at the 300-ft. level has been drawn, and the orebody is shown in the shaded area. The electrical phenomenon is evidently not centered on the section of the orebody. This is because the general axis of the deposit is not vertical. It outcrops approximately in the vicinity of the negative center, and dips steeply toward the southeast of the map, under the swamp.

The fact that the apex of the orebody lies under the negative center has been further demonstrated by sending an electric current directly into the orebody by means of an insulated wire lowered into the drill hole *A* (Fig. 2), which encountered the mineralization at about 150 ft. Inasmuch as the ore possesses a conductivity that is almost infinite in comparison with that of the surrounding rocks, the entire mass of the orebody is energized at about the same potential, and its exterior is an equipotential surface. Other equipotential surfaces are somewhat parallel to the first one. By tracing equipotential curves at the surface, the center of emergence of the electrical current was determined. This point corresponds approximately to the position of the apex of the orebody.

Fig. 2 shows the equipotential curves obtained by this work, centered around point *O*, the point of emergence of the current. Note that this point is only 50 ft. from the self-potential center *C* (Fig. 1), thus leading to the same practical conclusion as to the position of the upper part of the conductive deposit. This experiment proves that the conductive material underlying point *O* is well connected electrically with the orebody encountered by drill hole *A*.

Of interest to note is the fact that Curve 3, which possesses the lowest potential, is of large size. It extends abnormally toward the west as far as the outcropping contact between the rhyolite and andesite. This demonstrates particularly well, in my opinion, the electrical continuity of the thin mineralized sheet that occurs at this contact.

The whole survey demonstrates that the *F* deposit of the Amulet Mines gives rise to an important phenomenon of spontaneous polarization (more than 250 millivolts). A deposit of this type is consequently amenable to the self-potential method of exploration.

The apparent discrepancy between the horizontal section of the orebody at the 300-ft. level and the center of the spontaneous electrical phenomenon at the surface is explained by electrical investigations. The conclusions thus reached are in accordance with geological information. In the area of points *O* and *C* is an upper part of the orebody, investigated by drilling and of small commercial value; this upper part, nevertheless, is a good conductor of current, and constitutes the apex of the whole electrically conductive mineralization. The apex is on the western side of the commercial mineralization, as could be expected, inasmuch as the deposit is dipping toward the east.

Such an example of field work shows how electrical indications given by the self-potential method should be carefully and thoroughly investigated before being considered useless. In this particular example, a geologist who would have put down merely a vertical diamond drill at center *C* would have missed a valuable orebody, and unjustly condemned the electrical method of exploration, in an application where results are logical and satisfactory.

Discovery of the Brunschwig orebody near Hope, B. C., was the result of an electrical survey carried out for the British Columbia Nickel Mines in 1930. The large body of pyrrhoite contained nickel and copper. Inclosing rock in this formation is a dike of peridotite.

The reconnaissance was conducted by means of self-potential profiles oriented north-south. These profiles are numbered *P.I* to *P.V.* on Fig. 3. The survey resulted in the rapid discovery of an area of strong spontaneous electrical activity, in the southern part of the territory surveyed. An equipotential curve *C* gives roughly the trend of the mineralization. Two diamond drills, bored in the vicinity of profile *P.* III, resulted in the discovery of a massive orebody about 55 ft. wide, of good grade nickeliferous and cupriferous pyrrhotite. A tentative cross-section of the deposit along the line *XY* is given on the figure. No surface indications suggested the presence of an orebody in this area. The electrical survey was completed in less than two weeks.

Two examples of practical surveys for ore by the self-potential method have been given. In both, the electrical conclusions agree satisfactorily with the geological information. Discussion of such concrete examples of field work should be beneficial to the mining fraternity in enabling the geologist and the mining engineer to understand better the practical use of electrical exploration, and to appreciate the services it may render for the study of a given problem.

Among the numerous geophysical methods which can be employed for the reconnaissance of mineral claims, the self-potential method is quick and cheap. It is also one of the most reliable, inasmuch as it detects only the metallically conductive masses buried in the ground. It gives, therefore, indications entirely independent of the electrical conductivity of the rocks, which is of an electrolytic nature.

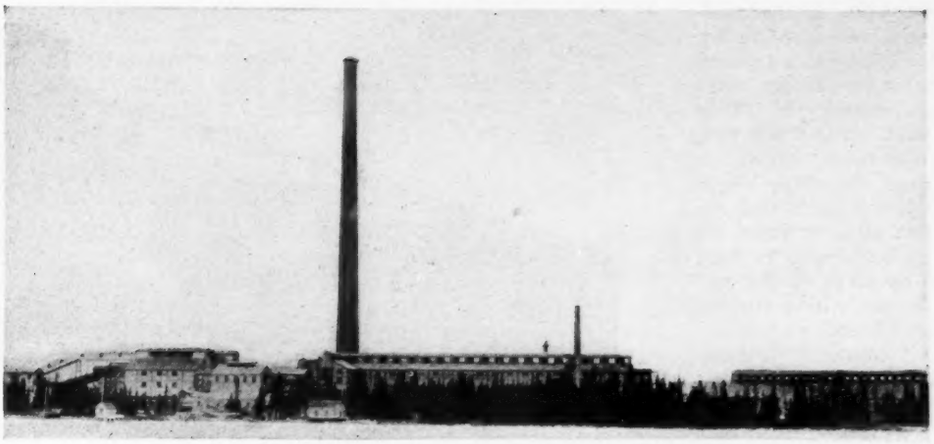


Fig. 1—View of plant, dominated by tallest stack in Europe

Smelting at Rönnskär, Sweden

Photos Courtesy of American-Swedish News Exchange

A. G. Paul Palén

AT Rönnskär, on a small peninsula projecting into the Bay of Bothnia, southeast of the city of Skellefteaa, a new smelter has recently been completed for treatment of the auriferous copper ore from Boliden, the latest and by far the most promising addition to the mines of Sweden. Good harbor facilities and proximity to the State railway system gave Rönnskär preference over other locations. Railroad connection between mine and smelter was completed in the autumn of 1929.

Plans for the smelter were begun in 1926; details were completed during the winter of 1927-28, and construction was started early in 1928, with work on the harbor and the smokestack. Foundations for the main smelter buildings were laid in the summer of 1928, and the construction work continued during 1929. In December of that year the crushing plant was put into operation. During the first part of 1930 the different furnaces were successively started, and the first copper from the converters was poured in June, 1930. About this time the development at the mine showed that a larger tonnage of ore was available than the first exploration had indicated; and a large increase in the plant capacity was therefore decided upon. As provisions for future expansion had been made in the first layout this could be done without interfering with the original flowsheet.

Smelting ore received from the mine is a mixture of arsenopyrite, chalcopyrite, and iron pyrite, with an admixture of small amounts of other minerals containing gold, silver, lead, zinc, and antimony. It has the following average analysis: Copper, 2.1 per cent; arsenic (As), 10.8; sulphur, 28.7; iron, 31.7; silica, 18.4 per cent; silver, 70 grams per ton; gold, 22.4 grams per ton. As the silica is not sufficient to slag the iron, barren quartzite from the hanging wall of the mine is used as flux. This will be replaced later, at least partly, by a siliceous ore carrying some gold and silver. Lime rock, shipped in from the Baltic island of Gotland, is also added.

The unique composition of the Boliden ore—especially its high arsenic content—necessitated careful study to work out a flowsheet that would give the best results, metallurgical and economic. After a series of comprehensive investigations, a treatment according to the flowsheet given in Fig. 3 was adopted, the principal steps of which are: Crushing and bedding of ore and fluxes; roasting to eliminate arsenic and part of the sulphur; removal of the arsenious oxide from the cooled roaster gases in Cottrell treaters; smelting of the calcines in reverberatory furnaces; blowing of matte to blister copper; preliminary refining of the blister and casting of anodes; electrolytic refining, with recovery of gold and silver; melting, refining, and casting the electrolytic copper for the market. In addition, units for recovery of byproducts, including an arsenic refining plant, are being planned.

The general arrangement of the plant is shown in Fig. 2. Ore from the mine is hauled over a trestle (1) to the bedding bins (3) via the crushing and sampling plants (2), if not already crushed at the mine. The ore mixture is elevated to the top of the roaster buildings (4), the gases from which are cooled and passed through Cottrell treaters (5) before entering the flue system leading to the stack (18). Calcines are smelted in the reverberatory furnaces (9); the matte is converted to blister copper, which is subjected to a preliminary refining in the anode furnace (14), and cast into anodes for the electrolytic refinery (15). The cathodes are smelted in the wirebar furnace (16) and cast into wirebars and other standard shapes for shipment. Coal for the reverberatory furnaces arrives by steamers and is unloaded at the harbor on the north side of the island. By building the coal-drying and pulverizing plant (17) in close proximity to the storage, the crane for unloading is also used for transportation of the coal from the storage to the pulverizing plant.

Crushing and Bedding—The ore, arriving from the mine in 25-ton bottom-dump standard railroad cars, is discharged into receiving bins below the trestle. As it has been crushed below 10 mm. ($\frac{3}{8}$ in.) at the mine, it is taken from the receiving bins directly to the bedding bins by a belt conveyor, being sampled on the way with a Morgaardshammers Kiruna automatic sampler. Occasional lots of custom ore, lime rock, and secondaries from the furnaces pass through the crushing plant, where crushing is in three steps—by jaw crusher, 24-in. Symons disk crushers, and by rolls; the crushed material is transferred to the bedding bins with belt conveyors. Materials such as cleanings and converter slag, which are charged to the reverberatory furnaces without roasting, go directly to the storage bins. A Kiruna automatic sampler is also incorporated in the crushing plant.

The bedding system consists of five bins of the free-hanging, parabolic type,

three of steel construction in one building, and two of steel and wood in another building. Each bin has a capacity of about 4,000 metric tons of ore, making a total storage capacity of 20,000 tons. On top of each bin is a belt conveyor with a movable tipper, which distributes the ore over the whole length of the bins in such a manner that the various constituents are arranged in horizontal layers. Reclaiming is done vertically through openings in the bottom. A movable hopper with a feeding arrangement discharges the ore mixture on a system of belt conveyors running to bins on top of the roasting furnaces.

Roasting—The first section of the roasting plant is equipped with four seven-hearth Lurgi furnaces, outside diameter of 6.5 m. (21½ ft.), and capacity 45 to 50 tons per furnace per 24 hr. Arsenic is almost completely removed in the roasting process, only 0.5 to 0.9 per cent remaining in the calcines. Sulphur is kept at about 8 to 9 per cent, giving a reverberatory matte with 15 to 20 per cent copper. In view of the increased capacity decided upon after completion of the first roasting furnaces, a larger furnace was found expedient for the extension. Each of the two following sections was therefore equipped with four eleven-hearth furnaces with a 7-m. (23¾-ft.) diameter and an individual capacity of 100 tons of ore per 24 hr. All furnaces have air-cooled rabble arms, the hot air being used to heat the bedding bins during the cold season. The rabbles are of special cast nickel-chromium steel, common cast iron being rapidly destroyed by the arsenic.

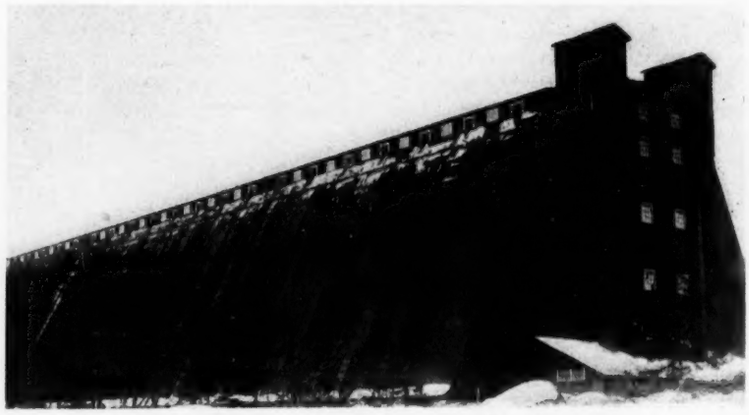


Fig. 4—Arsenic storage, capacity 120,000 metric tons

One of the biggest problems in planning the smelter was the removal of the large amounts of arsenic from the roaster gases, as no similar experience on a large scale was known. The gases must first be cooled below the condensation point of arsenious oxide (218 deg. C.). Later experience has shown that cooling must be carried below 150 deg. C. to secure good efficiency in the Cottrell treaters. The first coolers were large chambers of sheet iron, equipped with air-cooled jackets to provide sufficient cooling surface. An experiment with water cooling did not prove successful. On the later coolers several improvements have been made, and the cooling problem may be considered satisfactorily solved. About one-third of the arsenious oxide is recovered in the coolers, the rest being precipitated in the Cottrell treaters, which are of the plate type, of Lurgi construction.

At present the arsenic precipitated in the coolers and Cottrell treaters is drawn into closed cars and dumped on a belt conveyor running in an underground tunnel to one of two elevators, which carry the powder to the top of the arsenic storage (Fig. 4), where another belt conveyor discharges it through slots in the roof. Transportation from coolers and Cottrells to the belt conveyor, however, will soon be effected by screw conveyors instead of cars. The arsenic storage is a large concrete structure, 110 m. long, 40 m. wide, and 23 m. high, with a cubical content of about 50,000 cu.m., and capable of holding about 120,000 tons of crude arsenic.

In addition to the multiple-hearth furnaces, a 2x24 m. (6½x79-ft.) rotary roasting kiln has been installed, which treats about 50 tons of ore per 24 hr. This installation, built according to the Gröppel patents, with provisions to introduce air at points along the kiln, is working highly successfully. For future expansion, the present plan provides space for another block of four eleven-hearth furnaces. More rotary kilns may also be added.

Reverberatory Smelting—Three reverberatory furnaces fired with coal dust are provided for smelting calcines. The one first built has a hearth area of 5x25 m. (16½x82 ft.) and is equipped with four burners; the two units completed later are 6x29 m. (19¾x95 ft.) and have five burners each. They are constructed according to American practice, of silica brick with a 20-in. arch in the hottest zone and 15 in. toward the slag tap. Charging is done from bins, to which the calcines are transported in closed cars on a bridge running above the furnaces. Converter slag is returned to the furnace through an opening in the side near the fire bridge. Slag is tapped in the usual manner, from the end opposite the fire bridge, and matte is drawn through three tap holes in the side, toward the slag tap. The slag is granulated or drawn into cast-iron

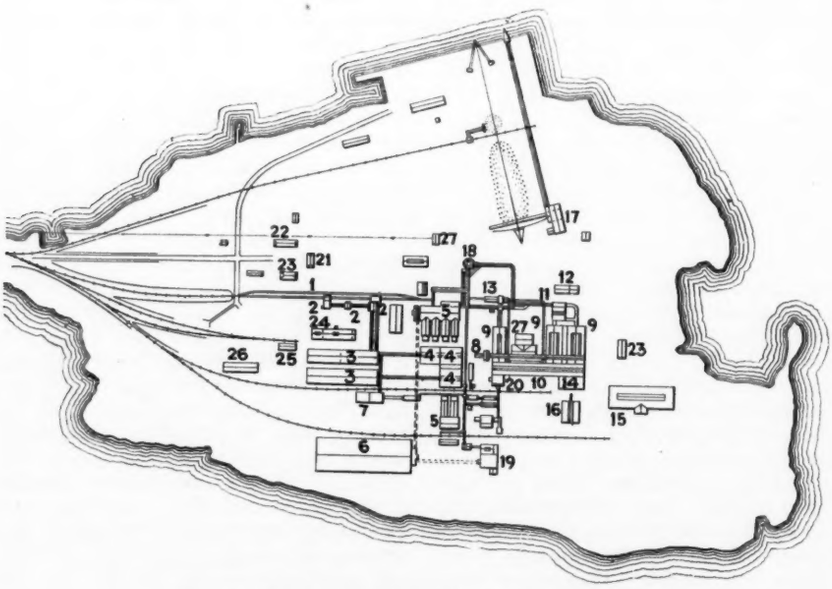


Fig. 2—General arrangement of plant, scale 1:2,000

1 Elevated ore receiving track
2 Crushing and sampling
3 Bedding bins
4 Multiple-hearth roasters
5 Arsenic precipitation
6 Arsenic storage
7 Rotary roasting kiln
8 Elevators
9 Reverberatory furnaces
10 Converter aisle
11 Boiler house
12 Power house
13 Cottrell treater
14 Anode furnace
15 Electrolytic refinery
16 Wirebar furnace
17 Coal pulverizing plant
18 Stack
19 Arsenic refining
20 Blast furnace
21 Office
22 Laboratory
23 Change houses
24 Machine shop
25 Electric and carpenter shop
26 Warehouse
27 Transformer stations

pots and dumped in liquid state. Change from granulation to liquid slag and visa versa is quickly made.

The coal burners have individual direct-current motors, permitting regulation of the speed according to the amount of coal required. Air for combustion is delivered from two ventilators, of which one is sufficient for the whole furnace, the second serving as a spare. Air to the individual burners is regulated with dampers. As stated, the coal is dried and ground in a central plant equipped with five "Rema" grinding mills. Pulverized coal is pumped with compressed air in 2½-in. pipelines to bins at the furnaces, from which it is fed to the burners.

The matte produced analyzes from 15 to 20 per cent copper, which allows a high gold recovery. A typical analysis of the slag is the following average for January, 1931: Copper, 0.21 per cent; silica, 36.9; alumina, 4.6; ferrous oxide, 46.0; and lime, 4.5 per cent. Nothing can yet be said regarding the ultimate capacity of the reverberatory furnaces, as the roaster capacity has not been sufficient to force them to the limit. Of the twelve multiple-hearth furnaces, only four of small capacity and two large ones are in operation, but more roasters will soon be started.

A boiler house with a 1,100-kw. de Laval steam turbine is under construction for the recovery of the waste heat from the two large reverberatory furnaces. Part of the steam produced will be used in the electrolytic refinery, and some for the distillation of water, as no supply of fresh water is available. Water is now transported to the plant in tank cars from the city of Skellefteaa.

Converting—In connection with the first reverberatory furnace, two small basic-lined converters 2x2.1 m. (6½x7 ft.) were installed, a reciprocating blower furnishing the blast. With the increased capacity these converters have now been replaced by three basic-lined barrel-type converters, 2.7x3.5 m. (9x 11½ ft.) shown in Fig. 5. The blast for these converters is furnished by two de Laval turbo blowers and one reciprocating blower, each with a capacity of about 125 cu.m. (4,400 cu.ft.) per minute, working with a pressure of about 1.2 kg./sq.cm. (17 lb./sq.in.). Matte and slag are transported in ladles of 1.2 cu.m. (43 cu.ft.) capacity, handled by two Halmstads Nya Verkstad traveling cranes of 15 and 25 tons' capacity on the main lift and 3 tons on the auxiliaries, of which there are two on each crane. The converter hoods are of sheet iron and are easily removed for barring out crusts. Gases and fumes are carried in a sheet-iron flue to a Cottrell pipe treater, Lurgi construction, where a dust is recovered carrying silver, gold, lead, and arsenic. Tilting of the converters is done by direct-current motors, supplied with power from a separate generator, connected with a storage battery as reserve.

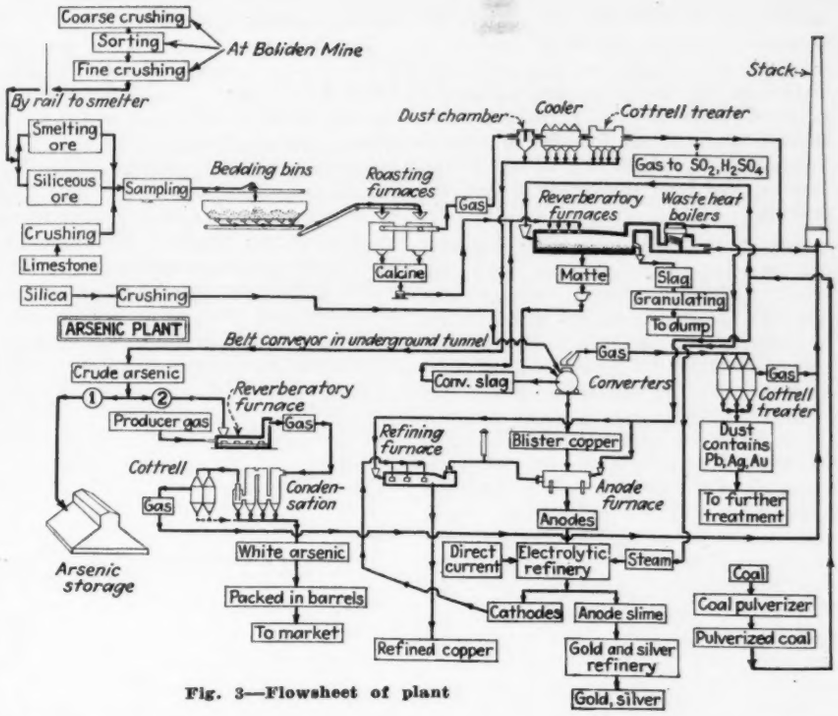


Fig. 3—Flowsheet of plant

Prior to the starting of the electrolytic refinery, the blister copper was cast in bars weighing about 90 kg. (200 lb.) and shipped for refining. Blister is now poured from the converters into a coal-dust-fired refining furnace where it is blown, poled, and cast into anodes. The anode furnace has a tilting arrangement similar to that of the converters. A coal burner is centrally placed at one end, with connection to flue and stack at the opposite end. Anode molds are placed on cars running on a track underneath the furnace, and pouring is done from a small ladle, which holds metal for two 125-kg. (275-lb.) anodes.

Electrolytic Refinery—The electrolytic refinery, built for an annual output of 6,000 tons, will probably be enlarged to twice that capacity in the near future. Special precautions have been taken on account of the high gold content of the anodes, which averages from 30 to 40 oz. per ton. For this reason the floor space is larger than that required in regular refining practice. Short anodes and wide spacing are used, and runways have been provided between every two rows of cells to permit easy inspection of all cells from the floor. Current density is lower than in usual practice. The cell room is served by two electric overhead cranes of 5 tons' capacity and Halmstads Nya Verkstad manufacture. Direct current for the electrolysis is produced by two Almänna Svenska motor-generator sets, 4,000 amps. at 100 volts, one of which serves a spare.

Connected with the electrolytic refinery is a special department for treatment of anode slime for recovery of gold and silver. A coal-dust-fired reverberatory furnace for smelting and refining cathodes is under construction.

Blast Furnaces—A blast furnace 1x3 m. (3½x10 ft.) at the tuyeres has been built for treatment of secondary products. It is equipped with tap

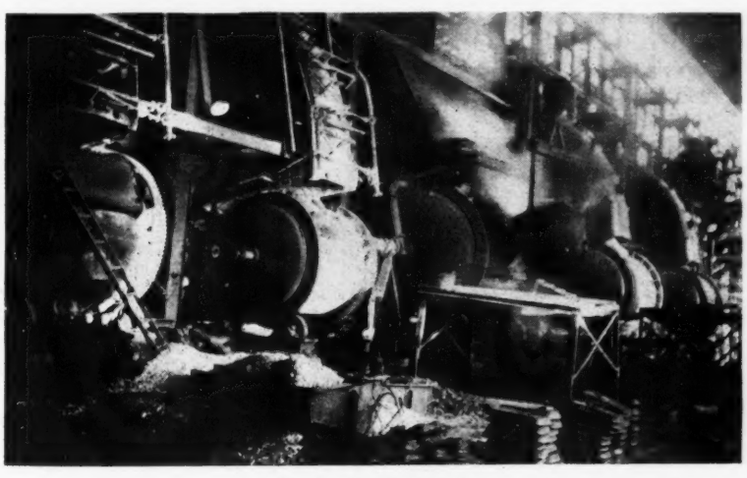


Fig. 5—Converter aisle

spouts at both ends, has two settlers for separation of slag and matte, and is so situated in relation to the converter aisle that the matte produced may be taken directly to the converters. A smaller blast furnace for special experiments, 1x2 m. (3¼x6 ft.) in size at the tuyeres, has been erected in a separate building.

Flue System—An extensive flue system connects each furnace with the stack. All flues, except the converter flue, are of brick with an outer casing of electrically welded sheet iron to prevent cooling of the gases by admixture of false air. By maintaining the gas volume as low as possible, the draft in the stack is not impaired. The stack is 145 m. (475 ft.) high, and has an inside diameter of 8 m. (26¼ ft.) at the base and 4.5 m. (15 ft.) at the top. It is constructed of a highly acid-resistant hard-burned brick. The hot gases from the reverberatory furnaces and the cooler gases from the roasters are carried in separate flues. Before entering the stack they pass through a system of mixing flues, where the temperature is equalized. Damage to the brickwork of the stack caused by fluctuations in temperature is prevented by the arrangement described.

Power—Power, delivered at 30,000 volts from power plants belonging to the city of Skellefteaa, is transformed to 1,500 volts in two stations (see Fig. 2, No. 27) and distributed to substations in the various departments of the plant, where further transformation to 220 volts for motors and 127 volts for the lighting system takes place. Larger motors, above 40 hp., operate on 1,500-volt current. The power plant operating on the waste-heat boilers serves as reserve.

Research Work—Extensive research work has been conducted to investigate the possibility of recovering byproducts. Utilization of the arsenic is of prime importance, and several new developments offer fair promise of a possibility to dispose of at least part of the arsenic production. As a first step in this scheme an arsenic refining plant has been designed and construction has been started.

General Arrangements—Other buildings on the site comprise a general office, a laboratory with special department for research work, a well-equipped machine shop, electrical shop, carpenter shop, and warehouse. Two change houses have been built, one in the western end, the other in the eastern end of the smelter site. The former, which is situated near the office, has special arrangements for the men employed in the arsenic department. It has a modern laundry for the free clothing provided these laborers. Altogether, the change houses accommodate about 350 men, and space is available for future expansion. Each has a spacious dining room, ample washing facilities, and electrically heated cupboards for the workers' food.

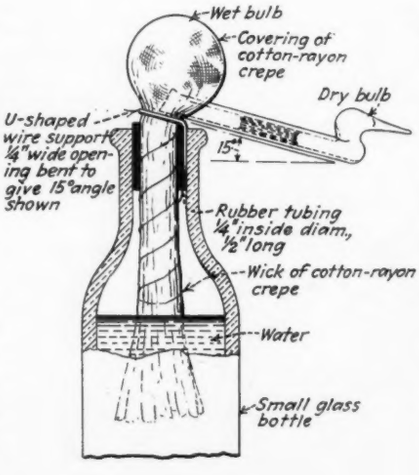
The laborers, drawn mainly from the local supply of forest and sawmill hands, have shown themselves well adapted to the new industry developed at Rönnskär. Only a few specialized furnace men and foremen have been employed from outside sources. Many of the men have their own homes within a few kilometers from the smelter; the remainder find accommodations along the railroad in the thickly populated district between Rönnskär and the city of Skellefteaa, a distance of about 20 km. Special trains run from Skellefteaa to the smelter, during change of shifts, afford adequate transportation facilities.

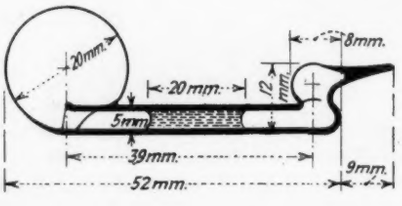
Some houses for the staff have been built on Hamnskär, adjoining the smelter site, but as the space there available did not prove sufficient, additional land has been purchased from the city of Skellefteaa, about 2 km. from the smelter. During the last few months fifteen houses have been erected here, accommodating about 35 families. A commodious mess house, with sixteen bedrooms and club facilities for bachelors and company guests, has been erected near the smelter.

■ ■ ■

Determining Conditions That Affect Comfort

HEREWITH is pictured a simple piece of apparatus that will indicate if the atmosphere in mine or plant or office is conducive to personal comfort and industrial efficiency. It consists of two bulbs, one small and one large, blown at the ends of ordinary glass tubing and fashioned as shown. The apparatus, closed from the atmosphere, contains a small amount of colored alcohol or other suitable liquid, which pulsates from one bulb to the other at a speed depending on atmospheric conditions. The action is explained as follows:


Wet bulb
Covering of cotton-rayon crepe
Dry bulb
U-shaped wire support ¼" wide opening bent to give 15° angle shown
15°
Rubber tubing ¼" inside diam., ½" long
Wick of cotton-rayon crepe
Water
Small glass bottle



The large bulb, inclosed in crepe that forms a wick dipping into water, preferably distilled, is cooled by the evaporation of water from its surface. The rate of cooling depends on three characteristics of the surrounding air—namely, its temperature, moisture content, and movement. Assuming the indicator liquid to be in the smaller bulb: the air in the larger bulb contracts as it is cooled, and draws the liquid up the inclined tube. At the conclusion of this phase of the operation, connection between the two bulbs is established in respect to air or vapor, and equilibrium restored. The liquid then flows down the inclined tube into the smaller bulb; whereupon the process is repeated. The greater the evaporation of water from the gauze around the larger bulb, the faster the pulsation of liquid between the two bulbs.

Operators and experimenters may decide for themselves what constitutes the most satisfactory index to adopt for any particular circumstance. Comfort conditions vary according to the amount of physical exertion expended, but 60 pulsations a minute has been taken as an average to indicate satisfactory atmospheric conditions. The determination may be simplified by using the apparatus in conjunction with a small sand glass, similar in design to the timer sometimes used to govern the cooking of eggs in hot water.

Construction of the apparatus shown calls for little additional comment. The wick, preferably of cotton-rayon crepe, should be a loose fit in the section of rubber tubing, so that air can easily enter the bottle to take the place of water evaporated. The maintenance of a definite angle of inclination of the tubing is important, and this is secured by the provision of the wire support shown.

Better Mine Ventilation And Lower Costs

Practice and results at the property of the Tom Reed Gold Mines Company, at Oatman, Arizona, demonstrate success from the application of engineering skill and the provision of adequate fan equipment in reducing operating expense per ton mined and improving working conditions

R. A. Elgin,
Safety and Ventilation Engineer
in collaboration with
V. A. Light,
Assistant Manager
and
C. R. Waters,
Mine Superintendent

THE Oatman district of Arizona was first prospected about 1862. In 1863 or 1864, John Moss is reported to have discovered free gold at what later was known as the Moss mine, and to have recovered about $240,000 from a pocket close to the surface. Mining was carried on in a desultory manner until 1902, when an important lode was discovered at Gold Road, about three miles to the north of Oatman. From 1902 until 1915, production was fairly continuous from the Gold Road mines. In 1901 the Tom Reed and Ben Harrison shafts were sunk 100 ft. on what later was to be known as the Tom Reed vein, which has since been worked by the Tom Reed Gold Mines Company from the Tom Reed, Ben Harrison, Grey Eagle, Bald Eagle, Aztec, Black Eagle, and United American shafts. Total production of gold from this vein has been about $15,000,000.

Present operations of the company are confined chiefly to the Black Eagle and United American properties. United American, although not owned by the Tom Reed, is under lease, and is connected to the Black Eagle on the 600, 950, and 1,100 levels. The relationship between the two mines is indicated on the accompanying ventilation diagram.

Oatman district is characterized by mild winters and hot summers. Temperatures of 100 to 108 deg. F., common during the summer months, are ameliorated somewhat by the dryness of the air and the aridity of the surrounding country. Annual rainfall is about 8 in., much of which occurs as torrential rains during the summer months. The excessive heat and humidity in the mines of the district have contributed largely to high mining costs. The story is told by Mr. Waters that men on the 1,400 level of the Ben Harrison mine had a standing bet that no one could drive a 60-penny nail through a 3-in. lagging without stopping to rest, and that the bet was never won. Despite conditions such as these, it was not recognized until recently that ventilation could be afforded by small mining companies. This realization resulted in the installation of the present system of ventilation control at the Black Eagle and United American mines.

Mining had been done from an old inclined shaft to a depth of 90 ft. below the 600 level at the Black Eagle between 1911 and 1924. Following this, leasers sank a winze to a point a few feet below the present 800 level. In 1927, decision was made to sink a vertical shaft, 4½x13¼ ft. in the clear, to the 1,100 level, and to drive a crosscut to a body of high-grade ore that had been topped below the 800 level. The shaft was raised from the 600 level to the surface, and sunk from that point to the 1,100 level. Despite a flow of between 200 and 250 gal. water per minute, at a temperature of 100 deg. F., the shaft is reported to have been sunk at a cost of $89 per foot. Two Cameron sinking pumps were used to handle the water. In later sinking operations a 14-in., 6-stage, Layne & Bowler pump, capable of handling 500 gal. water per minute against a 200-ft. head, was used when a heavy flow of water was encountered.

The first 200 ft. of the 1,100 crosscut was driven without great difficulty, but at this point a flow of 1,100 gal. water per minute was encountered. Water poured from the face and back of the drift in sheets, and rendered ventilation by a No. 4 Buffalo blower entirely inadequate. The flow of water was so great that it was not uncommon for a section of track, after being laid, to go floating down the drift.

The vein was struck after crosscutting 551 ft. The flow of water continued as great as ever, but the temperature had risen to 103 deg. F. A drift was driven along the vein a distance of 222 ft., after which connection was made with the 950 level through raises from the 1,100 level and a winze from the 800 level.

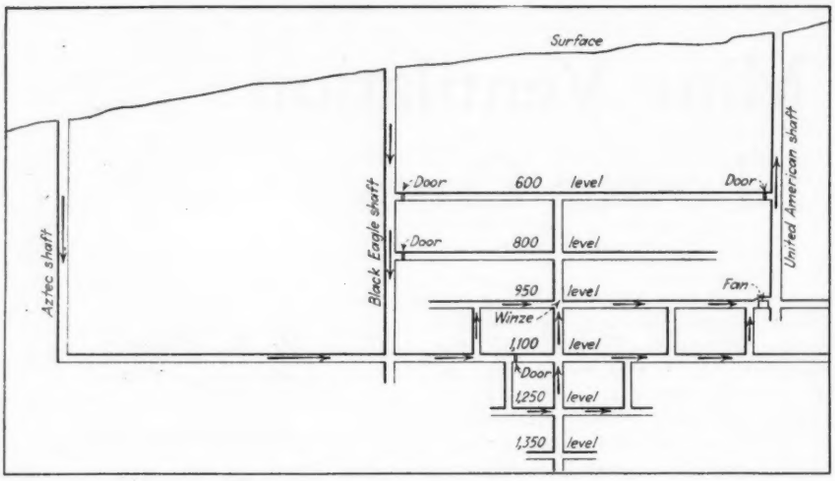
Until this time, ventilation had been entirely inadequate. With mine air near saturation point, and at a temperature close to 103 deg. F., positive ventilation was necessary to insure efficient mining operations. It was estimated that a No. 8 Sirocco, driven by a 25-hp. motor, calculated to deliver 31,000 cu.ft. air per minute against a static pressure of 2½ in., would fulfill the needs of the mine. This fan was ordered and installed on the 1,100 level near the shaft station, and a great improvement was noted in the mine air. However, development was continued, laterally and horizontally, on the 950 and 1,100 levels. A connection was driven to old workings in the Aztec mine, which were themselves connected with the Aztec shaft. Meantime, the 950 level had been driven to the United American, and connection made with the United American shaft.

In 1931, decision was made that the ventilation system must be redesigned and the fan moved to a new location to obtain efficient operation. New development and greater extension of the mine workings made this necessary. The organization of a department with the double duties of ventilation and safety was established, which was expected to pay for itself.

An examination of the mine showed that the following conditions had developed:

	Temperature, Deg. F.	Relative Humidity, Per Cent
Intake air at collar of shaft	81	64
800 drift near station	88	98
800 drift at top of winze	89	97
950 drift	90	98
1,100 drift at station	90	74
Aztec drift	93	96
1,100 drift at winze	92	89
1,250 drift	98	100
Mine water	103	...

The air at all the working places had reached a high temperature, and was near its saturation point at that temperature. Air for the Aztec drift and for the 1,250 level was being supplied by two No. 5 Sirocco, high-speed blowers driven by 5-hp. motors. Air was delivered to the working faces through 16-in. metal pipe. Intake air on the 1,100 level would have given almost the same temperatures and relative humidities throughout the mine, regardless of initial temperature and relative humidity, because of the heavy flow of hot



Ventilation System in the Tom Reed gold mine

water along the drifts. The mine pumps were handling 350 gal. water per minute, all of which came in contact with the intake air at various points.

It was decided to move the fan to the bottom of the United American shaft on the 950 level and to install a system of control doors. Fan and motor were moved to their new locations and installed on concrete foundations. A rectangular steel évasé was built, together with a steel door on an angle-iron frame, and the two pieces were installed in an 8-in. concrete wall. The edges of the steel door were lined with old belting, which permitted little leakage. The American shaft was only occasionally used; so frequent opening of the door was unnecessary, otherwise necessity would have arisen for two doors, one ahead of the other, because of the strong pressure of the air against the door, making its opening difficult while the fan was in motion. The total cost of moving the fan and installing it in its new location was $515.

The control doors were of wood, built of two layers of cross-diagonal 1-in. boards, with a layer of old canvas between. They were framed on both sides with 1x6-in. strips, to prevent sagging. The door frames, of 4x6-in. timber, consisted of two posts, a cap, and a sill. Eighty-penny nails were driven along the outside of the frame, which was then anchored in a 6-in. concrete wall. The door was hung on ⅜x1-in. strap hinges; and, when hung, was inclosed in a 2x6-in. outer frame, nailed to the 4x6-in. inner frame. The doors were comparatively light, but proved to be very strong; none has been replaced since installation. Doors are kept closed by a weight suspended from an old piece of tugger cable, fastened to the door and threaded through a pulley. Cables are fastened to the door with a small eyebolt. Pulleys are suspended from a stull or from an eyebolt driven in a plug in the back of the drift.

The average door required about 108 bd.ft. of lumber, 2½ sacks of concrete, and 10 sacks of sand and gravel. The cost of the average door as installed is estimated to be $30. Both sand and gravel are obtained locally from near-by stream beds.

Upon completion of the new system, the underground atmosphere was found to be as follows:

Place	Temperature, Deg., F.	Relative Humidity, Per Cent
Intake air	74	56
800 drift near station	89	54
800 drift at winze (dead air)	86	90
950 drift at stopes	84	80
1,100 drift at stopes	85	76
1,100 drift United American	92	78
1,250 drift	89	85

Although the mine is still not cool, the better distribution of the air and the reduction in its relative humidity at the working faces have made possible continuous work in reasonable comfort. The relative humidity of the intake air is not representative, inasmuch as rain fell on each of the days when the examinations happened to be made. The mine has continued to become cooler as time went on, because of absorption of heat from exposed surfaces. Temperature of the mine water in the 1,350 sump has now dropped to 87 deg.

The time and money spent in controlling the underground ventilation have paid for themselves. This is shown in the following table of mine costs, in which the cost of breaking, mucking, tramming, and ventilation is included:

A 5x7-ft. drift, *without adequate ventilation:*

		Per Foot
1,100 drift — First 555 ft.		$14.10
Next 222 ft.		17.25
Next 81 ft.		15.66

The same drift, *with adequate ventilation:*

	Per Foot
Next 778 ft.	$10.79

Not to give ventilation an unfair advantage in the comparative costs, the section of 81 ft. at $15.66 is compared with the 778 ft. at $10.79. At the time the 81 ft. was driven, the water had subsided to a lower level, and was being pumped from the 1,100 winze. The only factor entering into the last figure that is not included in the first is adequate ventilation. The character of the ground was the same; only the last 778 ft. had the additional tramming distance. All tramming was done by hand.

A comparison may also be made in stope costs. The two stopes compared were in the same orebody, of the same hardness, and of the same dimensions. Both stopes averaged 5 ft. from hanging wall to footwall and were about 150 ft. long. Both were mined by cut-and-fill stoping:

No. 5,200 stope, 900 level, *not ventilated:* 22,392 tons mined at a labor cost of $3.11 per ton.

The same stope, 1,100 level, *with adequate ventilation:* 40,362 tons mined at a cost of $2.54 per ton, for labor and ventilation. The saving is shown as a labor benefit, inasmuch as the other stope costs were unaffected. *The saving in this one stope alone more than pays the entire cost of the ventilating system, including the cost of the fan.*

On the 950 level, a drift was driven in the United American in the same ground as the drift on the 1,100 level. This drift was well ventilated and shows the following interesting comparative costs:

Distance driven, ft.	257
Breaking	$868.84
Mucking and tramming	711.00
Ventilation	150.17
Total cost	$1,730.01
Cost per foot	$6.73

The tramming distance was about 1,000 ft. The drift was somewhat favored by being free from water.

Acknowledgment is made of U.S.G.S. Bulletin 397, by F. C. Schrader; and U.S.G.S. Bulletin 743, by F. L. Ransome, for information regarding early day mining in the Oatman district.

Transporting concentrate in Malaya

MINE CAGES

Lucien Eaton

IF THE waste produced by development on the lower levels of a mine is to be used as filling on the upper levels, cages afford a convenient means of hoisting it. This consideration, under certain circumstances, might justify the use of cages for all hoisting, as where a mine is producing a relatively small tonnage of high-grade ore in several different grades that must be kept separate, and a large proportionate production of waste. In a few operations cages are still used in shafts of large capacity, but they find their best application as an adjunct to skips.

Cages may be operated in separate compartments, or, if this is impossible, they may be hung above or below the skips. The latter arrangement has advantages in a small shaft, but also several weak points. It provides a convenient method of transferring waste from one level to another and for sending men and materials underground during the shift, but as a substitute for a man and supply cage in a separate compartment it has little in its favor. The extra weight of the cage added to that of the skip decreases the proportion of live load per trip, and increases the strain on the rope and the pressure of the coiled rope on the drum of the hoist. Furthermore, the cage must be of small capacity, because it has to pass between the plates of the skip dump, and, if it is hung above the skip, the height of the headframe must be increased to provide the necessary clearance. The practice of hanging the cage below the skip is also subject to criticism, because skip gates cannot then be used and the cage is always dirty from skip spillage. Reduction in headroom required in the headframe is offset by the necessity for greater depth in the shaft. In this arrangement the skip must, of course, be provided with safety catches.

The most serious objection to combining skip and cage arises from the operating difficulties it causes. The same objection can be made to using a skip and cage in balance. Too much time is lost by the ore-handling crew, waiting for the skips, when timber and other supplies are being sent down, and, also, by the timber-handling crew,

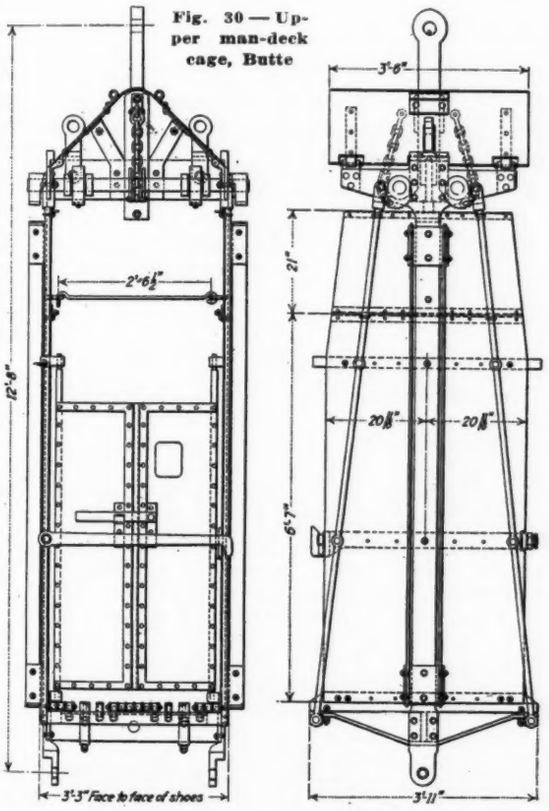


Fig. 30 — Upper man-deck cage, Butte

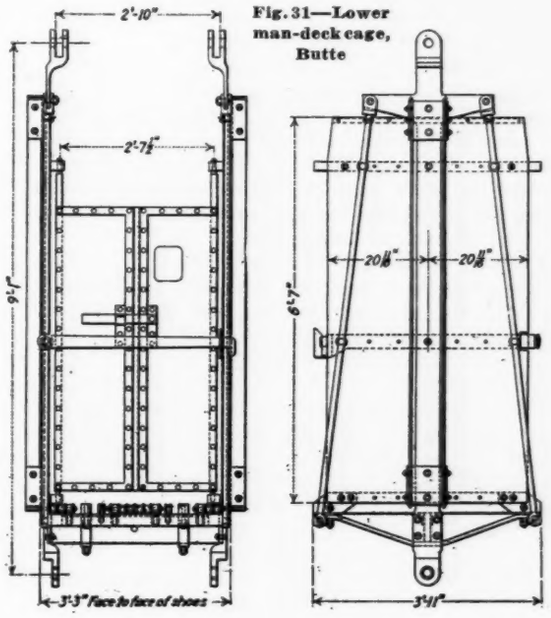


Fig. 31—Lower man-deck cage, Butte

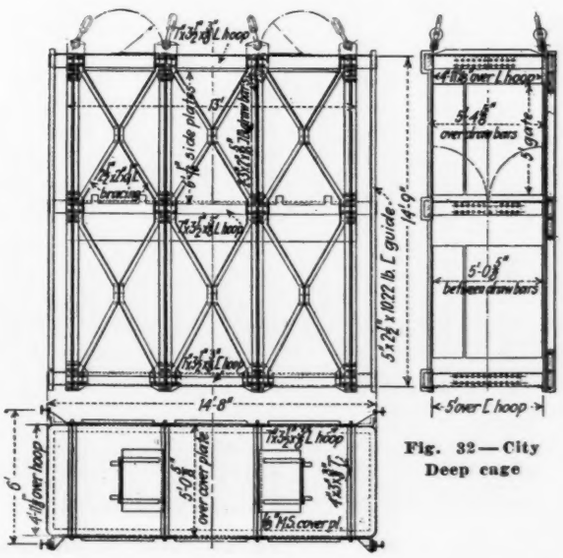


Fig. 32 — City Deep cage

This is one of a series of articles on "Mine Equipment."

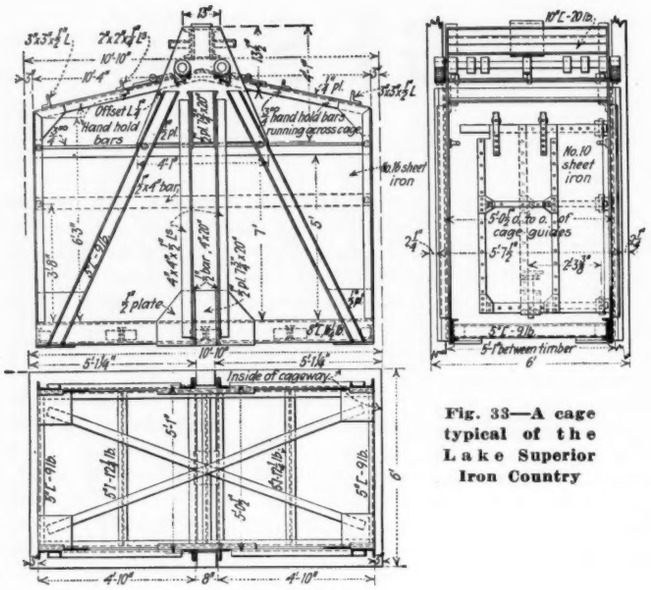


Fig. 33—A cage typical of the Lake Superior Iron Country

waiting for the cage while ore is being hoisted.

Cages used only for hoisting cars need not be entirely inclosed. Usually they have bars across the ends, instead of doors, to keep the cars in place. However, during the day, men are almost sure to ride the cage, and safe practice demands complete inclosure. The doors needed for protecting the men need not interfere with car handling, if they are so designed that they can be folded back against the sides of the cage. Cages of this design, used in Butte, Mont., are shown in Figs. 30 and 31, taken from an article by C. M. Harrer and L. S. Farnham, *E.&M.J.*, Vol. 129, No. 11, 1930. These cages are used in strings of three to five, hung one below the other. The upper cage only is provided with safety catches and bonnet, but every cage is completely inclosed with sides and doors of $\frac{1}{8}$-in. solid plate. Punched plates or even heavy screen might be used instead.

crosshead and sideframes of the upper cage are heavier than those below, but the platforms, sides, doors, latches, and guide shoes should be standard on all. In "changing" cages of this type they must, of course, be suspended from above, and provision for this is made on the upper crosshead.

The large cage shown in Fig. 32 is

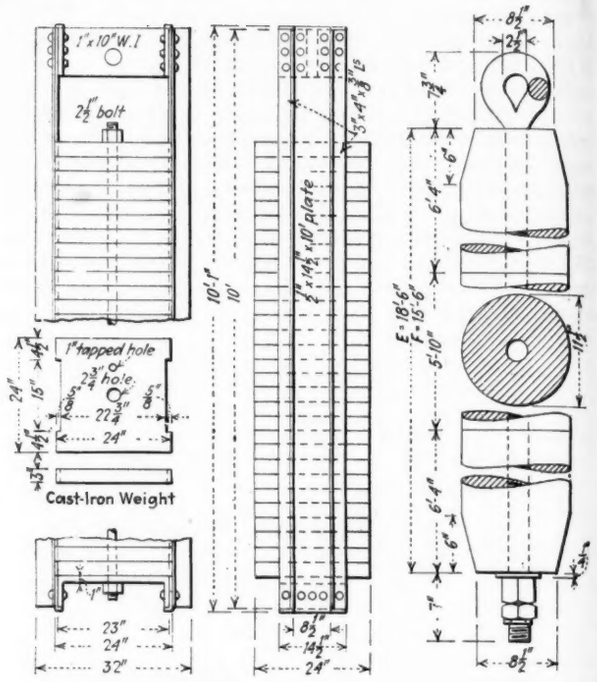


Fig. 35—Cast-iron counterweights used in two Lake Superior mines. Dimension E is the length used for 6,000-lb. weight, F for 5,000 lb.

The lower cages, all alike, are suspended one from the other by "eyes" and bolts, as shown. This arrangement has much in its favor for multiple-deck cages. It permits a high degree of standardization, and also saves weight, because the only parts on which the load is cumulative are the sideframes. The

used in the City Deep mine, on the Rand, for hoisting ore in an 8-ton car on the lower deck ("Winding at Great Depths," by W. Elsdon Dew. Third Empire Mining and Metallurgical Congress, South Africa, 1930). No safety catches are provided. The cage is hung by eight chains, to prevent the hoisting

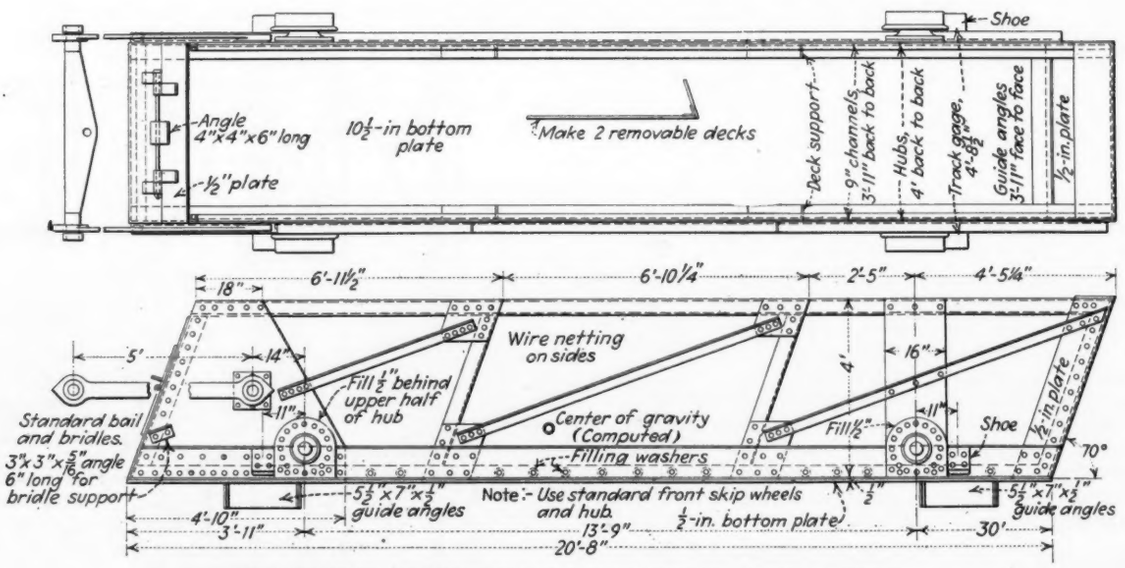


Fig. 34—Design for three-deck man car, Champion Copper Company

rope from kinking, in case of too much slack. Four 80-lb. rails serve as guides. The man capacity is large, each deck carrying 55 men.

A design typical of the Lake Superior Iron Country is shown in Fig. 33. Only one cage is used, run in balance with a counterweight. This cage is seldom used for hoisting rock, its function being to handle men and supplies. Twenty-eight men can ride on the one deck, and the body is large enough to carry a 6-ton locomotive or a 4-ton car. Timber up to 9½ ft. long is carried horizontally on trucks. This cage is completely inclosed with 16-gage sheet iron, and at each end has a large door, which swings inward against the side. Each door has a sliding panel. Handholds and a car stop are provided. The crosshead is built substantially and carries safety catches and a hinged bonnet. The bottom framework is hung from the crosshead by the side plates and by diagonal angle-irons.

In selecting the design of a cage several precautions should be observed. The first is to see that the drawbar, when in its lowest position, does not come down far enough to strike the head of a man riding on the cage; and the second is to have the central, stationary part of the bonnet overlap the movable, hinged parts, as one shingle overlaps another on a roof. This will keep the cage dry and comfortable even in a wet shaft. The doors should be built so that they cannot open outward; and, if wide, they should have sliding panels, and should abut against permanent stops on the floor near the center line, so that they cannot be sprung outward beyond the edge of the cage by pressure from within. Latches should be designed so that they can be opened from either inside or outside. Handles or bolts should not project beyond the clearance line of the cage.

If metal guides are used, safety catches will not work successfully, some even insisting they cannot be depended upon with wooden guides. Properly designed dogs, if kept in good condition, seldom, if ever, fail to catch. I prefer dogs of the normal involute shape, like those shown in Fig. 33. These have grooves with sharp edges between them cut across the face that comes in contact with the guide. Three double rows of pins about 2 in. apart, in addition to these grooves, are a great help. The pins are of ⅝-in. round iron, flattened to a chisel edge at one end and threaded at the other, and are screwed into holes in the face of the dog until they stick out about 1 in., leaving the chisel edge set across the grain of the guide. These teeth will penetrate ice formed on the guides, or the outer soft layer of wood that is sometimes found in wet shafts. The shafts on which the dogs are mounted must turn freely, and their bearings should be of ample size. Lubrication of these bearings is too often neglected. Coil springs that take their load under compression, and are of ample length and diameter, give satisfactory service and are easily inspected or replaced. They should always be in duplicate.

Cages for Inclined Shafts—In deep inclined shafts, such as those in the Michigan Copper Country, cages or man cars are customarily substituted for the skips at the beginning and end of every shift. An example of the cage used in one of the steep shafts, that of the Champion Copper Company, is shown in Fig. 34. (Bureau of Mines, Information Circular 6515, 1931.) This cage has three decks and is inclosed on three sides. Safety catches are not used, because of the absence of guides or back runners. In the flatter shafts the man car is open, and is arranged like a flight of stairs with steps, on which the men sit.

In a timbered inclined shaft, a cage equipped with safety catches may be used if back runners or guides are installed, upon which the dogs can bite. Such installations are rare, but work satisfactorily.

Counterweights for skip or cage are not often used in inclined shafts in the United States. In vertical shafts, however, they have become popular in connection with man-cages. A large man-cage balanced by a counterweight takes no more room than two smaller cages in balance, has nearly the same hoisting capacity up to a depth of 1,500 ft., and is more convenient to operate, especially when serving more than one level.

The rope from the counterweight is sometimes carried over a head sheave and attached directly to the cage, but is more often wound on the hoist drum opposite the cage rope. The first method is not recommended, because the two ropes may become entangled.

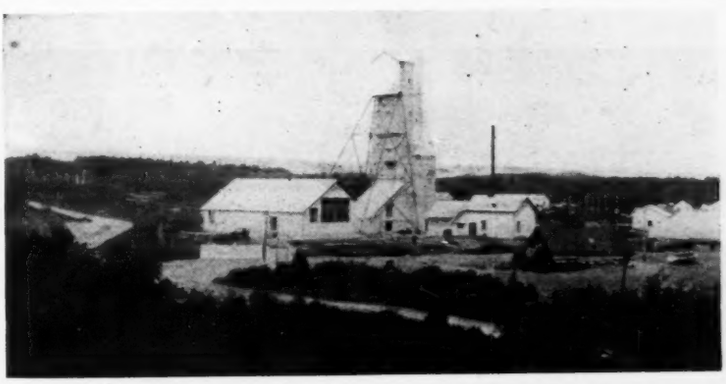
Counterweights may run on guides or in pipes. Simple designs of both kinds are shown in Fig. 35. Large counterweights, weighing more than 6 tons, are usually of the A type and consist of a frame with guide shoes, which carries a loaded box or cast-iron or concrete blocks. Guides should preferably be the same as those used for skip and cage, but this is not essential, because, if the guide shoes are long and the gage is narrow, wear on the guides is almost negligible.

A counterweight up to 6 tons in weight can be conveniently run inside a 12-in. pipe. One is shown at *B* in Fig. 25. It consists of cast-iron sections 11½ in. in diameter strung on a 3-in. wrought-iron rod, which has a welded eye at the top and a locked nut at the bottom. Waste oil and grease is dumped into the pipe occasionally for lubrication. The one disadvantage of this arrangement is that ropes which spin, such as those of ordinary Lang lay construction, cannot be used in this type of installation.

Chairs—Customarily the cage is supported on chairs, when loaded cars are put on or off. In very deep shafts, where the stretch of the rope is important, the use of chairs is almost essential. This is true also if the hoist drum is not fast on its shaft. If it is fast, however, chairs are unnecessary when handling timber and other supplies or men; in fact, the spring of the rope is of assistance in removing loaded trucks from the cage.

Chairs may be mounted directly upon the cage, in which case they consist of four sliding bars, resting upon the shaft timbers and moved in or out by a hand lever on the cage. Or they may be mounted on the shaft timbers at each level, in which event they consist of four short arms, mounted just below the level on two shafts, which are rotated by levers arranged so that each arm supports a corner of the cage. This is the usual construction.

The disadvantage in the use of chairs is that, if inadvertently left in place, they may cause a wreck. A simple and effective type of chair not open to this objection consists of a short chain attached to a timber in the shaft in such a position that it can be swung out to engage a downward-pointing hook on the side of the cage. When the cage is hoisted, the chain is automatically released, and falls back out of the way. Instead of the chain, a hinged hook can be used which will catch under the edge of the cage. This device supports the cage on only one side, but the shaft guides serve to prevent it from being thrown out of alignment.



Teck-Hughes, one of Canada's large gold mines

Why Substitute Synchronous Motors for the Induction Type?

THE advantages gained by substituting the synchronous motor for the induction type in the driving field arise primarily from the ability of the former to send into the lines to which it is connected a certain amount of exciting kilovolt-amperes required by induction motors operating on the same lines, to raise the power factor of those lines, and to reduce the load upon lines, switches, transformers, and generators (as stated by R. S. Sage in a recent issue). Under certain circumstances, a saving is effected, due primarily to obviating the need for additional investment in the equipment mentioned, and to reducing current consumption in these parts of the system. In some operations, a saving also results from the higher operating efficiency of the synchronous motor, and, if a company is buying power, adoption of this type of motor may result in a reduction in the power rate.

One of the largest metal-mining companies uses synchronous motors for driving all new machinery, if the cost is justified and the drive can be handled by a motor of this type. Ordinarily, this operator does not substitute synchronous motors for induction motors already installed, as its power contract does not contain a penalizing clause for power factor below 80 per cent. The synchronous motors in this case are used on blowers, air compressors, motor-generator sets, and ball mills. The company's most recent application of this type of drive is on ball mills, where a 200-hp., 164-r.p.m. motor is used to drive a 7x10-ft. ball mill. This mill runs at 20 r.p.m. and is driven through a single reduction, whereas the old mills are driven by a 200-hp., 720-r.p.m. induction motor through a double reduction. Here a material saving is evidently effected, not only on the first cost of gears but on maintenance as well. An induction motor designed for 164 r.p.m. would not be economical with respect to first cost or space requirement.

At the mine and mill of a large zinc company, the motor load is approximately 3,000 kw., or between 4,000 and 4,500 hp. Of this 2,200 hp. is in synchronous motors, which are said to have netted a saving of $10,000 a year. These motors range from 50 to 600 hp. and are used on air compressors, rotary converters, and pumps. This company is buying its power under a power factor contract, and the power factor for its total load will average between 92 and 95 per cent. The superintendent holds that the greatest economy in using motors of the synchronous type is in the lowered repair bill. Most of the motors have required little or no repairs and are as good as new, he says, after eighteen years' service. This is not true to the induction motors, however, the stators of which in many instances have had to be rewound because they have been allowed to run too close, such rewindings having had to be done two or three times under certain conditions. For this reason, he thinks, the extra first cost of the synchronous motor is justified when a motor must be bought. Aside from the lowered repair cost, the saving to the average small consumer of power is slight, and will remain so until the power companies are willing to allow the consumer to benefit to a greater extent from the resultant economies of operation.

In those plants where the engineering department has long been familiar with the characteristics of the two types of motors, the types most suitable for each particular drive will probably have been selected, so that today no opportunity exists for effecting a saving by changing the motor installation. For example, all the plants of one company have a power factor of 85 to 90 per cent, so that the electrical engineer has never had occasion to substitute synchronous motors to maintain the most economical power factor. This engineer says, however, that where power cost is high and power factor low, the replacement of induction motors with over-capacity synchronous motors for power-factor correction would be profitable.

■

Novel Safety-Record Board Arouses Interest

AN EFFECTIVE METHOD to stimulate interest in maintaining a low accident rate at the Hillsboro plant of the Eagle-Picher Lead Company that has insured cooperation among employees, as well as arousing favorable comment from others, is reproduced below. Each paling of the fence bears the name of an employee, the entire working force at the plant being represented. Protection from a colored individual, labeled "Accident," is provided by the fence for a row of chickens, each of which bears the name of one of the departments of the plant. A rooster, labeled "Yards," representing the department having the best safety record at the time the photograph was taken, leads the row of chickens.

Additional information supplied by Eagle-Picher concerning the safety record of the Hillsboro plant is to the effect that (1) no lost-time accident occurred in April, when the number of men employed averaged 84; (2) a total of 10,482 man-shifts has been worked without an accident so far this year; (3) the last accident at the plant occurred in June, 1929.

THE EAGLE-PICHER LEAD CO.
SAFETY RECORD
HILLSBORO PLANT
BEST PREVIOUS RECORD 558 DAYS
DON'T YOU BE THE WEAK PALING IN OUR SAFETY FENCE
PRESENT RECORD 1046 DAYS

Analyses of Iron-Ore Shipments From Lake Superior Ranges

With Ratios of Each Grade Specified and the Tonnage Produced

ACCOMPANYING TABLES give the average analysis of iron-ore shipments made in 1931 from the six principal ranges of the Lake Superior Iron Country, as recently announced by the Lake Superior Iron-Ore Association. Three of the ranges—the Mesabi, Vermilion, and Cuyuna—are in Minnesota. The others, in Michigan, are the Gogebic, Marquette, and Menominee ranges.

Iron content is expressed in terms of so-called "natural" iron, which is the percentage of iron in the ore before the latter is dried for analysis and which therefore must be determined by calculation from the dry analysis.

Five classes, or grades, of ore are specified: bessemer; low-phosphorus non-bessemer and high-phosphorus non-bessemer, according to the phosphorus content (the limits of which for each class are given in the tables); manganiferous; and siliceous. Table VII shows the percentage of each of these classes produced on each range.

The total tonnage for the six ranges was 23,281,333 in 1931, compared with 46,698,554 tons in 1930 and 65,443,546 tons in 1929. Of the 1931 tonnage, the Mesabi Range shipped 15,219,155 (42,657,601 in 1929), the Vermilion 1,129,189 (1,855,013 in 1929), the Cuyuna 807,436 (2,438,140 in 1929), the Gogebic 2,879,091 (7,547,957 in 1929), the Marquette 1,798,695 (5,393,939 in 1929), and the Menominee 1,447,767 (5,550,896 in 1929).

Generally speaking, the average tenor of ore shipped annually from each range has changed little in the last ten years (the period covered by the Association's report). This is due to the practice of "ore grading," whereby ores of different analyses are mixed, prior to shipment to Lower Lake Ports, so as to yield lots or cargoes of desired standard grades. The ability to maintain these grades is dependent to a varying extent on the practice of ore beneficiation, particularly in shipments from the Mesabi. Beneficiation, as practiced, may be limited to crushing and screening, which is the treatment given to over 70 per cent of the beneficiated ore, or may consist of washing, sintering, drying or jigging, the last three being practiced thus far in only a few instances.

Table I—Bessemer Ores

(*Phosphorus 0.45 per Cent or Under*)

Range	Iron (Nat.)	Phos.	Silica	Manganese	Moisture	Per Cent of Total Tonnage for Range
Mesabi	54.06	0.038	8.58	0.38	9.55	30.1
Vermilion	58.13	0.042	6.18	0.11	6.78	71.0
Cuyuna	.....	.....	....	....	....	....
Gogebic	54.91	0.040	7.20	0.36	10.15	18.6
Marquette	54.62	0.036	9.85	0.16	9.37	8.2
Menominee	.....	.....	....	....	....	....
Total bessemer	54.69	0.039	8.17	0.33	9.23	26.0

Table II—Low-Phosphorus Non-Bessemer Ores

(*Phosphorus 0.180 per Cent and Under Down to 0.045 per Cent*)

Range	Iron (Nat.)	Phos.	Silica	Manganese	Moisture	Per Cent of Tota
Mesabi	50.22	0.072	8.31	0.84	12.51	68.7
Vermilion	60.06	0.090	6.06	0.09	3.94	29.0
Cuyuna	53.35	0.081	11.68	0.22	9.88	12.7
Gogebic	52.30	0.076	7.25	0.61	12.45	75.2
Marquette	52.37	0.102	8.12	0.49	10.17	65.7
Menominee	50.24	0.072	8.39	0.30	7.18	36.1
Total low-phos. non-bessemer	50.94	0.075	8.11	0.74	11.92	63.4

Table III—High-Phosphorus Non-Bessemer Ores

(*Phosphorus Above 0.180 per Cent*)

Range	Iron (Nat.)	Phos.	Silica	Manganese	Moisture	Per Cent of Total
Mesabi (*A*)	.....	.....	....	....	....	....
Vermilion (*A*)	.....	.....	....	....	....	....
Cuyuna	52.54	0.219	9.83	0.65	8.81	34.8
Gogebic (*A*)	.....	.....	....	....	....	....
Marquette	47.26	0.285	10.36	0.32	11.89	8.9
Menominee	52.11	0.450	6.57	0.35	7.63	57.6
Total high-phos. non-bessemer	51.60	0.379	7.76	0.41	8.42	5.5

Table IV—Manganiferous Ores

(*Manganese 2 per Cent and Over*)

Range	Iron (Nat.)	Phos.	Silica	Manganese	Moisture	Per Cent of Total
Mesabi	50.73	0.109	8.81	3.11	8.62	1.1
Vermilion (*A*)	.....	.....	....	....	....	52.5
Cuyuna	39.81	0.250	9.60	8.31	11.85	
Gogebic	49.34	0.068	8.04	4.68	9.95	6.2
Marquette (*A*)	.....	.....	....	....	....	....
Menominee	46.71	0.591	7.22	3.90	7.37	5.9
Total manganiferous	44.65	0.218	8.88	6.08	10.37	3.7

Table V—Siliceous Ores

(*Silica 18 per Cent and Over*)

Range	Iron (Nat.)	Phos.	Silica	Manganese	Moisture	Per Cent of Total
Mesabi	39.90	0.039	33.80	0.37	6.00	0.1
Vermilion	.....	.....	....	....	....	....
Cuyuna (*A*)	.....	.....	....	....	....	....
Gogebic	.....	.....	....	....	....	....
Marquette	37.40	0.049	41.31	0.09	2.90	17.2
Menominee	36.59	0.019	41.30	0.06	3.20	0.4
Total siliceous	37.46	0.048	41.07	0.09	3.00	1.4

Table VI—All Grades, Total Tonnage

Range	Iron (Nat.)	Phos.	Silica	Manganese	Moisture
Mesabi	51.37	0.062	8.42	0.73	11.57
Vermilion	58.69	0.056	6.14	0.10	5.96
Cuyuna	45.96	0.218	9.95	4.62	10.55
Gogebic	52.60	0.070	7.29	0.82	11.87
Marquette	49.52	0.104	14.18	0.38	9.01
Menominee	51.05	0.320	7.42	0.54	7.43
Grand total	51.53	0.087	8.60	0.80	10.84

Table VII—Classes of Iron Ore Produced on Lake Superior Ranges in 1931

Range	Bessemer Ores, per Cent	Low-Phosphorus Non-Bessemer Ores, per Cent	High-Phosphorus Non-Bessemer Ores, per Cent	Manganiferous Ores, per Cent	Siliceous Ores, per Cent	Total, per Cent
Mesabi	30.1	68.7	(*a*)	1.1	0.1	100
Vermilion	71.0	29.0	(*a*)	(*a*)	....	100
Cuyuna	....	12.7	34.8	52.5	(*a*)	100
Gogebic	18.6	75.2	(*a*)	6.2	....	100
Marquette	8.2	65.7	8.9	(*a*)	17.2	100
Menominee	....	36.1	57.6	5.9	0.4	100

(*a*) Production statistics for the years 1922-1931 inclusive, show no recorded tonnage of the above class of ore for this range.

Among Contributors To This Issue:

Lucien Eaton (Harvard) is in London. He served for several years with the Cleveland-Cliffs Iron Company before appointment as consulting engineer to Rhodesian Selection Trust.

✦

Owen Letcher (Redruth School of Mines) is a mining engineer-journalist, Johannesburg correspondent for *Engineering and Mining Journal*, for many years, and well known as a writer on South African affairs.

✦

Morris J. Elsing (Columbia School of Mines) was formerly geologist to Copper Queen Consolidated, later in charge of the geological department and of the mining operations of Cananea Consolidated, to which company he subsequently served as consulting engineer.

✦

Russell B. Paul (Colorado School of Mines), who contributes the article on "Welfare and Safety at Gilman, Colorado," is assistant general manager of mines of The New Jersey Zinc Company. Prior to transfer to New York, Mr. Paul was general mine superintendent for The Empire Zinc Company in Colorado.

✦

H. D. Keiser (Lehigh and Pennsylvania State College) has served on the technical staffs of the Bureau of Mines and the Geological Survey of the United States, in addition to gaining experience in mining operations in Australia, Africa, and the Americas. He joined the editorial staff of *Engineering and Mining Journal* in 1927.

✦

J. B. White is an honor student at the Mackay School of Mines, Reno, Nev. The house described in the article on a successful application of non-metallic minerals for dwelling insulation belongs to and was designed by **Jay Arnold Carpenter,** professor of mining at the school, widely known in Western mining circles as an operator and engineer as well as an educator.

C. O. Stee, who describes a series of successful metallurgical operations in Quebec, was graduated from the University of North Dakota in 1911, and has since gathered a broad technical and geographical experience in the practice of his profession. He has served in many operating and executive capacities on the staffs of Cerro de Pasco, Peru; Butte & Superior, Montana; Idawa Gold Mining, Idaho; Moctezuma Copper, and Cia. Real del Monte y Pachuca, Mexico. Since 1930 he has been manager of the Siscoe property.

✦

Dr. L. H. Duschak (Michigan and Princeton) has specialized in chemical and metallurgical research. Between 1913 and 1921 he was in charge of the Pacific Experiment Station of the United States Bureau of Mines, at Berkeley, Calif. The collaborator with Doctor Duschak, in the research described and in the preparation of the article on the use of starch in flotation, **D. N. Vedensky,** is research metallurgist on the staff of the Treadwell Yukon company.

✦

A. G. Paul Palén, metallurgist to Bolidens Gruvaktiebolag, was graduated from the Stockholm School of Mines in 1904. He has since been connected with various metallurgical enterprises in Sweden, Norway, and the United States, including Anaconda and Old Dominion. In 1926 he was appointed in charge of the planning, construction, and operation of the metallurgical plant at Rönnskär, Sweden, described in this issue.

✦

J. B. Huttl, until recently assistant editor of *Engineering and Mining Journal,* was educated at the M.A.N. Technical School and the Stadische Bauschule, Nurnberg, Germany. He obtained operating experience with mining companies in the Ruhr district of Germany, in Spain, Mexico, and Central America, specializing in mechanical engineering and plant maintenance.

An $80,000 clean-up at Siscoe, Que., Canada. Left to right: Owen Matthews, mill superintendent; C. O. Stee, the manager; and Joe Lagotha, the amalgamator

C. W. Boise has returned to London from the United States.

A. J. McNab, recently in Northern Rhodesia, has returned to London.

R. W. Karpinski recently left Ann Arbor, Mich., for Alaska, on mine examination work.

Dr. C. K. Leith has been nominated for the presidency of the Geological Society of America for 1933.

R. D. Smith, formerly with Hampden Cloncurry Mines, is now with the Day Dawn Company, New Guinea.

H. C. Chellson has returned from Russia after completion of a contract with the U.S.S.R. as consulting geologist.

F. C. Carstarphen was invested on May 20 with the degree of doctor of engineering by the Colorado School of Mines.

D. A. Cheyette is now in charge of the Chicago office of the Traylor Engineering & Manufacturing Company, at 1 La Salle Street Building.

Richard Hamilton was recently elected president of the Chamber of Mines of Western Australia for the thirty-second successive annual term.

Dr. Anton Gray has gone to Serbia, Yugoslavia, to undertake geological work for Trepca Mines and other Selection Trust companies.

Keith Cameron, formerly mine foreman at Mount Isa, Queensland, is in charge of the Comstock mine of the Mount Lyell company, Tasmania.

C. H. E. Stewart is at Great Bear Lake, North Canada, supervising development and exploration activities. He expects to return to Toronto in September.

Dr. W. H. Collins, director of the Geological Survey of Canada, an authority on Pre-Cambrian geology, has been elected a member of the American Philosophical Society.

Frank Ayer, manager of Morenci Branch of Phelps Dodge, was presented with a Joseph A. Holmes Safety Association Certificate of Honor at a public meeting sponsored by the Miners' Club of Morenci, at Morenci, Ariz., on May 19.

Dr. David White, senior geologist of the United States Geological Survey, has been awarded the Mary Clark Thompson gold medal of the National Academy of Sciences, for distinguished achievement in paleontology.

Gerald G. Dobbs, general superintendent of the ore mines and quarries of the Tata Iron & Steel Company, of India, has been granted six months' leave and plans to visit the United States and Canada in the near future.

Scott Turner, director of the U. S. Bureau of Mines, and **C. Harry Benedict,** metallurgist on the staff of Calumet & Hecla, will each receive the honorary degree of doctor of engineering at the forthcoming commencement exercises at the Michigan College of Mining and Technology.

Mining Geology And Mine Exploration

The Editor:

The abstract from an article by A. D. Hoffman in the March issue of *Engineering and Mining Journal* speaks of the predominance of articles written by college professors and students of mining geology, and directs attention to the fact that too few who have had experience have published their findings. As a result new discoveries are rare and the cost of exploration has increased.

The college professors perhaps may be excused for not attempting to develop a new technique in the art of ore finding, as their field experience is necessarily limited. However, in a perusal of reports written by many mining geologists, one is struck by the purely academic nature of the context. In spite of this, many sound and practical ideas on the genesis of ore deposits have been gathered and published during the last few years by mining geologists with wide field experience. Undoubtedly, much valuable information has been lost because experienced men have not published their findings. Unfortunately, most practical field geologists are more efficient with the pick than they are with the pen.

The cost of prospecting work is bound to increase as time depletes the present known reserves, many of which discovered themselves, in that they at least gave some indication of their presence on the surface or underground. This condition is rapidly becoming a thing of the past, except in virgin fields, and is keeping step with the exit of the old-time prospector. A greater number of valuable ore deposits lying buried under a few feet of soil and rock up to the maximum depth of mining still await discovery. Most past discoveries were confined to the present surface.

The future demands the combined effort of mining geologist, geophysicist, and practical miners. A new technique will be developed in which the mining geologist will be required to remain long enough in the field to prove his theory. Although a sound theory put into practice often leads to success, failure is frequently due to the attempt to apply the same rule or theory to areas not favored by the same geological conditions. The high cost of prospecting is the result of the constant reexamination of the same area. If the work is properly done at the beginning, results will generally yield reliable information at low cost. The present practice cannot be considered thorough. It frequently does not include a close study of the purely mechanical features of the invaded rocks. This phase of the work will require, and should receive, more attention in the future.

If the mining geologists could obtain but a small part of the funds squandered each year by boards of directors and technical committees on propositions of questionable merit, some of the perplexing problems relating to the formation of ore deposits would be solved, and ore reserves increased. After world conditions return to normal and international trade again assumes its natural flow, the demand for metals should far exceed the present maximum possible output. Future discoveries will depend on the combined efforts of the practical geologist, geophysicist, and miner.

Brussels, Belgium. DUNCAN SMITH.

Floating the first airplane-transported gold dredge at Bulolo, New Guinea, Nov. 14, 1931

Cooperation Is a Privilege

The Editor:

With reference to Mr. S. D. Strauss' article on gold production in the May issue: The biggest high-grade gold mine in the world was overlooked—the Tom Reed, of Oatman, Ariz. Production was over one million dollars in 1931. The ore averaged over $30 per ton.

Give honor where honor is due!

Santa Barbara, Calif. "DESERT RAT."

[Mr. Strauss admits that our correspondent is correct. Statistics of the Tom Reed company were omitted, as were also those of five other companies, all of which produced sufficient gold to merit inclusion in the list. All these companies were approached for data, and either refused to comply or neglected to answer letters of inquiry. To those who cooperated to permit so nearly complete an analysis, honor is due and accorded.—EDITOR.]

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Obituary

G. McMillan Ross, at Stockton, Calif., April 29; aged 81. Mr. Ross was at one time assayer at the United States Mint at Carson City, Nev.; and later superintendent of the Consolidated Virginia and Ophir mines.

F. E. Grosvenor, fourth Baron Ebury, at London, May 15; aged 48. Lord Ebury, after attendance at the Camborne School of Mines, practiced his profession in the Isle of Man and in Canada, specializing in metallurgical engineering. He served at various times on the directors' board of several Canadian mining companies, and earned a distinguished war record.

William Prellwitz, chief engineer, Ingersoll-Rand Company; April 12, at Easton, Pa.; aged 73. Of his qualities an associate writes: "Many abilities were combined in 'Pop' Prellwitz—inventive genius, an appreciation of sound engineering principles, good business judgment, much common sense, and an unusual capacity for working harmoniously with his fellow men. He contributed much to the advancement of compressed-air machinery, more especially to the development of the products of the company with which he spent his entire adult life."

J. F. Borden, at Hawaii, from heart failure, March 31. Mr. Borden graduated from Cornell in 1904, acting for two years thereafter as instructor in the College of Engineering, University of California. After general mining experience, he joined the staff of Oliver United Filters in 1915, specializing in metallurgy and sugar technology.

A. M. Plumb, at Kansas City, Mo., March 21; aged 59. Educated at Cornell University and the Colorado School of Mines; spent thirty years in mining in Colorado, Montana, and Wisconsin; invented the Plumb pneumatic jig, used in the treatment of high-gravity ore. For sixteen years, Mr. Plumb was employed by American Zinc, Lead & Smelting, of Boston, and was manager of the company's properties at Platteville, Wis., for ten years.

B. K. Emerson, emeritus professor of geology, Amherst College; at Amherst, Mass., April 7; aged 88. Professor Emerson was one time president of the Geological Society of America, and assistant geologist for the U. S. Geological Survey. A friend says of him:

> "Professor Emerson had very striking characteristics and possessed a keen eye, always with a twinkle in it. At several of the American Association meetings he was the most active member in Section Q, which gave the familiar "wise-isms." At one time when he was intensely concentrated upon his work, he began some microscopic work in bright daylight on a Saturday—afternoon. Suddenly he was aware of bells ringing, and thought that there must be a fire. It was, however, Sunday morning and the bells were the church bells ringing for Sunday services. He had looked through the microscope all night. When a letter appeared without an address in the post office it was generally Professor Emerson who had mailed it. In arranging to purchase a large collection of minerals under consideration by Amherst College, which he was to have bought at a certain date, he was overjoyed to find that by not having purchased the collection on the proper day he had saved $500. In this instance he promptly forgave himself. Some twelve years ago when Professor Alfred B. LaCroix, Secretaire Perpetuelle de l'Academie des Sciences, visited the United States while the British Association meetings were in session, and the late J. Pierpont Morgan gave a sum of money which financed this trip to the United States, Professor La Croix, Professor Emerson and I had the pleasure of going through certain points together. Professor Emerson had all his notebooks with him, and these were as neatly written as though they had been done the previous day. They were so clear that they could have been reproduced directly from the written page."
>
> G. F. K.

Mining a Non-Metallic Mineral With a Coal Cutter

Dipl. Ing. Barth
Flottmann, A. G.
Herne, Germany

REPLACEMENT of hand methods of mining rock salt by introduction of mechanical devices was attempted in 1928 in a government-owned mine on the Continent. A Flottmann rotary bar cutter, of the Westfalia type, was used to cut at 5-m. intervals in a vertical face, to a depth of 1 m. The cutter was driven by an 11-kw. motor, operating at 380 volts, 21.5 amp., 1,450 r.p.m., and 50 cycles. Results proved that maximum capacity was attained with crude salt almost free from contamination. Salt mixed with anhydrite, found to be the most difficult material to cut, is worked easier and more economically by hand methods. Whereas the cutting speed under the most favorable conditions is 1 sq.m. in 6 min., in salt mixed with anhydrite this may decrease to 0.5 sq.m. in an 8-hr. shift, during which period up to 400 bits may have to be changed in the cutter bar. Two men are required for the operation of the cutter, one of whom starts the machine and regulates the speed of cutting; the other keeps the cut clean with a specially designed shovel.

Total net cutting time is unfavorable on account of the mining method in use, which necessitates the moving of the cutter to a new location almost every shift. Moved by its own power the cutter attains a speed of about 25 m. per hour. During this operation it is placed on a 15-cm. high truck, running on the mine tracks. Normal capacity of the cutter, operating in salt fairly free from contamination, is 4 to 5 m. per hour, after allowing for time lost in moving, but not for the time required in changing the bits on the cutter bar—an operation taking about ½ hr. and involving pulling the bar out, removing old bits and putting in new ones, and inserting the bar. In addition to the actual operation of the cutter, about one shift per day is required for servicing work such as sharpening dull bits, laying electric cable, and moving the main switch.

In 1929, two additional machines, with stronger drive housing, were purchased. Comparing the amount of salt mined by hand methods and that mined with the cutter, the following saving was accomplished by the latter method during the first nine months of 1930, figuring an amortization of 15 per cent and taking into account cost of power, repair parts, and similar expenses: January, 24.59 per cent; February, 24.72; March, 33.11; April, 40.40; May, 25.10; June, 31.70; July, 32.86; August, 37.55; and September, 58.60 per cent. Consumption of repair parts, such as bits, and of lubricants, totaled about 700 crowns ($20.75).

The money to be saved by the use of a cutting machine depends on the quality of the salt and the nature of the beds. For horizontal beds the operation is much more expensive than for vertical or steeply inclined beds. This explains the great saving in September, when some horizontal beds were cut. At one of the shafts of the mine 40-50 per cent salt is mined with machine; at the other shaft, 80 per cent salt. In the first of these machines, cutting cannot be introduced on a large scale because of the variable elevations of the working places and the comparatively small cutting lengths obtainable for this reason.

The following results were obtained during the month of September, 1930: Total number of shifts worked, 102; actual cutting time, 160 hr.; total cutting surface, 585.6 sq.m.; time used in moving cutter, 69 hr.; time used in preparatory work, 146½ hr.; number of bits changed, 4,370. For the other months similar results were obtained.

Hand cutters, receiving 20-25 per cent higher wages than machine operators, break about 0.8 sq.m. per shift. To cut the 585.6 sq.m. completed in 102 shifts by machine, hand operators working by contract would require 700-750 shifts.

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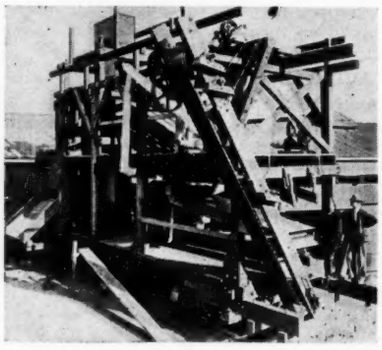
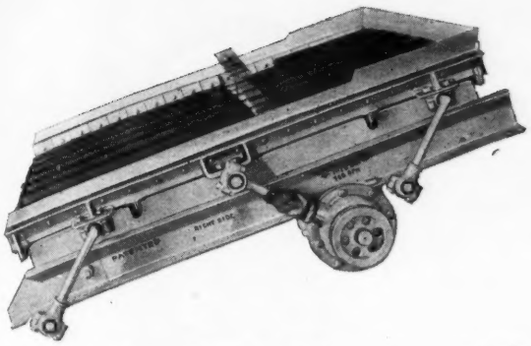
Gold-Concentrator Developments

Some months ago, the Cottrell Engineering Company, of Los Angeles, Calif., constructed a portable dry placer "dredge" for the Panamint Mining Company. The dredge is mounted upon railroad trucks; weighs 28 tons, and is powered with a 60-hp. gasoline engine. It is in effect a dry-washing unit and is used with a ¾-cu.yd. caterpillar-mounted gas shovel. The shovel discharges into a hopper upon the bottom of which is a grizzly with ¾-in. bars spaced 1 in. apart. Oversize is discharged to the side and undersize is fed to the boot of a bucket elevator which discharges upon a double-deck vibrating screen, the top screen having a ⅜-in. opening, and the bottom a ⅛-in. opening. The products are treated dry upon shaking sluices equipped with riffles. Fine material is treated upon an inclosed shaking sluice in which an air current is forced from beneath and suction applied above the bed. Cocoa-matting is used for the porous bed and cross riffles catch the heavy minerals. The dredge is 32 ft. long and 9 ft. wide.



Portable dry placer "dredge" built for Panamint Mining Company

In addition to the foregoing, the company manufactures a de-waterer,



This shaking screen has a sectional cover and all joints protected by rubber "wrists"

classifier, and sand washer of the rake type and makes special screens for the non-metallic industry.

Another Gold Concentrator

Among equipment developed for treating gold gravels and free-milling ores is a so-called "gold placer unit," introduced by Denver Equipment Company, Denver, Colo. As designed, this combines a grizzly and a shaking screen, followed by a shaking sluice launder, having riffles, cocoa matting, and amalgamation plates, with a Denver Sub-A flotation machine as the final step. This combination of flotation with gravity separation and amalgamation makes it possible to recover fine gold as well as flour gold from feed as coarse as $\frac{1}{4}$ in., which, moveover, can be handled without choking. The sluice box is removable. From it the feed goes to a mercury trap for recovering any mercury, amalgam, or gold which may escape from the launder.

New Vibrating Screens

Two new types of vibrating screens have been introduced by Link-Belt Company, Philadelphia. One is for heavy duty, having either single or multiple decks. It has a so-called positive drive, the amplitude being fixed at the factory to suit the work to be done. The other is also for heavy duty, but it is of the unbalanced-pulley type, intended for high-speed work where the material is sticky or for close sizing where the screen openings are small.

Another Vibrating Screen

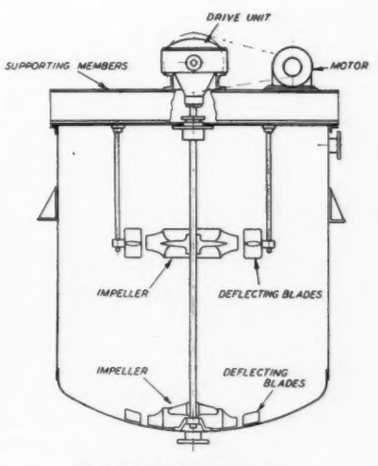
A new vibrating screen, having a rigid screen body with one or more decks given a vibration of fixed amplitude by the rapid throw of an eccentric shaft, has been introduced by Stephens-Adamson Manufacturing Company, Aurora, Ill. It has three units, the screen body, vibrating mechanism, and subframe. The vibrating mechanism is self-contained and consists of an eccentric shaft, adjustable balance wheels, SKF bearings, grease seals, and rigid tubular housing. Adjustment is made in the factory so as to give the vibrating frequency and amplitude that will prevent blinding or excessive bouncing. The panels are so made as to permit quick replacement. The screen is hung on four cables at the desired screening angle, and is free to vibrate. Bumper springs are eliminated.

For Handling Sand, Crushed Stone and Abrasives

A portable trough conveyor having an 18-in. belt and intended for handling sand, gravel, crushed stone or any non-metallic mineral in small sizes has been introduced by the Fairfield Engineering Company, Marion, Ohio. Steel skirt plates run the full length and make a 6-in. trough. The carriage wheels are 48 in. in diameter and are mounted on roller bearings. The head pulley is 8 in. in diameter, lagged.

Changes in Recording Instruments

Innovations in design mark a new line of graphic recording instruments introduced by Esterline-Angus Company, Indianapolis, Ind. Spring clocks driving the chart are removable and interchangeable. All springs in a clock are wound at once by a crank instead of a key, and from the front. This makes possible arrangement of the instrument so that the chart-driving rolls can be coupled together, which synchronizes the chart accurately. Another development permits accurate recording of quantities which change rapidly. When the values measured are steady, the meter operates at a slow chart speed. When sudden rapid changes are to be recorded, the chart automatically speeds up to 3,600 times the slow speed, and, like a slow-motion film, records the rapid changes which then take place, later resuming the slow speed if necessary. Twenty-two different instruments are included in the line.



Essential features of Turbo-Mixer

Sectional Cover for Shaking Screen

A shaking screen having all joints protected from dust by means of rubber wrists and using a screen cover made in either 4-in. or 8-in. sections, instead of the usual continuous sheet, has been developed by David Cole, Mills Building, El Paso, Tex., who calls it the "rubber wrist" screen. This screen makes 900 full strokes per minute, each stroke imparting to the screen surface a positive movement of not less than one-quarter inch over its entire area in a direction at 45 deg. to it. Power requirements are about $\frac{3}{4}$ hp., which is usually applied through a short-center V-belt. Where the eccentric rods emerge from the motor case, the openings are closed by socks to prevent entry of dust or other substance. The screen cover panels are made of high-carbon, spring-steel wires and fit between 4-in. clips on either side of the frame. One bolt in each clip serves to draw them tight. In the phraseology of the inventor, the screen "kicks the oversize into the air, while hopping it down the slope." Thus the oversize is not permitted to slide upon and abrade the wires. When holes wear through the screen surface, only the sections affected need be replaced and not the entire cover. Tested at an Arizona concentrator, a section was found to last four months.

In the Field of Agitation

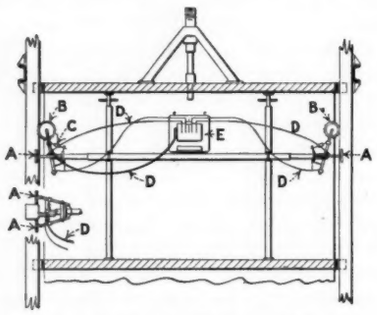
Arrangements have been made by The Dorr Company with the Turbo-Mixer Corporation for sales rights to the Turbo-Mixer and other Turbo equipment in certain of its principal fields. Compared with the Dorr agitator, which has found application chiefly where the time element in processing is 30 minutes or so, and where the solids and liquids must be kept in intimate contact and in motion with respect to each other during this period, the Turbo-Mixer has been used principally where an intimate mixing is required in the shortest possible time, with or without aëration. For conditions where an even diffusion of a large volume of air or gas throughout the contents of a tank is desired during agitation, the Turbo-Aërator, a variation of the Mixer, has been used.

The Turbo-Mixer consists essentially of a tank, a mixing mechanism, and a driving unit. It is shown in an accompanying cut. The tank may be of any shape or size, open or closed, under pressure or vacuum, and with or without heating or refrigerating coils. The mixer may be operated either continuously in series with other processing equipment or intermittently with batch loadings. A distinctive feature is the directing of the impeller discharge along radially outward lines by means of curved blades on the deflecting ring, so that virtually all of the kinetic energy of the liquid is utilized, once the con-

tents have been set in motion. This equipment is being used in cyanidation of gold ores. Owing to its rapidity, it is expected to find use at flotation plants for conditioning the feed with various reagents and keeping it in a homogeneous, non-segregating condition in the surge tanks ahead of the cells. It is also expected to be useful for agitating pulp ahead of vacuum filters, and for repulping cake with solution in double or series filtration. Likewise, in leaching oxidized copper slimes, it may be used for acidulating the slimes with sulphuric acid, if of acid-resisting construction, and even for actual leaching in a series of continuous agitation tanks ahead of the counter-current decantation plant.

Checks Condition of Shaft Guides

Safety in skip or cage hoisting is dependent among other things on the condition of the guides. If these become so worn as to increase the span between them, or ground movement decreases it, rough and jerky travel or derailment may result, or the skip or cage may stick in the guides, with resultant damage. Moreover, if guides



Apparatus for checking condition of shaft guides

become worn beyond a certain point, safety devices may be unable to function as they should. A device, known as the Adams-Fuess apparatus, for ascertaining and recording the condition of the guides throughout the length of the shaft has been developed by R. Fuess, of Berlin-Steglitz, Germany, and New York. Its nature is indicated in the accompanying sketch.

Essentially, this apparatus consists of a frame attached to skip or cage and traveling between the guides, the frame carrying at each end three rollers which contact with three respective faces of the guide on that side. These rollers, shown at *A* and *B* in the sketch, are held against the guides by spring contacts *C*, and any irregularities (over a certain minimum) in the faces are communicated through flexible members, *D*, to respective recording pens which enter their records on a paper chart on cylinder *E*, attached to the body of the skip. Through a reducing device, the graphic record of any irregularity is made just one-half of the actual amount of the irregularity. Of the five pens, two record the widths of the guides, two the distance between the guides and their respective guide shoes, and one records any increase or decrease in distance between the guides.

Testing Wear and Lubricant

To purchase lubricants wisely, and without complete dependence on the lubricant salesman, the buyer must be able at least to make certain simple tests. This calls for suitable apparatus, hitherto not available save for the complicated equipment found in laboratories. Recently, the Timken Roller Bearing Company, Canton, Ohio, has developed a so-called "lubricant and wear tester," which makes it possible to determine the load-carrying capacity of lubricants, to measure friction, and to determine the wear characteristics of any material simply and accurately. If necessary, the machine can be used for only one of these purposes. With it can be determined not only the lubricant best suited for any given equipment, but the lubricants purchased may be tested to see that they measure up to specifications. Inasmuch as the subject of lubrication is generally linked intimately with that of friction and wear, the production of a machine able to make tests in all three of these fields has definite advantages.

Hopper Cars of Aluminum

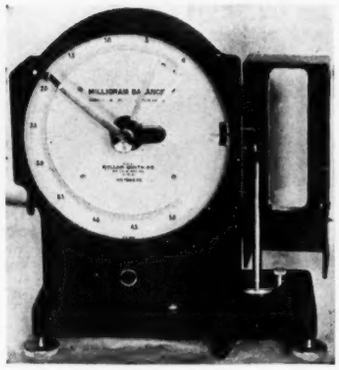
One of the subsidiaries of the Aluminum Company of American—namely, the Alcoa Ore Company—has been using for the last half year 70-ton copper cars with bodies made of corrosion-resistant aluminum alloy for hauling bauxite ore, coal, and other materials. The body is 40x9½ ft. inside, over-all length is 42 ft., and height above rail 10½ ft. Steel and iron are used in the running gear. Thus built, one of these cars weighs 21,200 lb. less than it would if the usual construction materials had been used, or 38,900 lb.

Novel Air Hoists

A portable hoist operated by a Hele-Shaw-Beacham air motor mounted inside the drum is likewise being made by Holman Bros. The motor has no connecting rods, cranks, or reciprocating valves. This hoist, known as the "Superhaul," is made in three sizes: HU, pulling 1,500 lb. at 100 ft. per minute; HR, pulling 2,500 lb. at 132 ft. per minute; and HO, pulling 3,000 lb. at 140 ft. per minute. The torque of this type of air motor is greatest on starting and does not fall off rapidly as the speed increases.

Speeds Up Precision Weighing

A precision assay balance said to permit a hundred weighings in a half hour or less in skilled hands has been introduced by Roller-Smith Company, 233 Broadway, New York. It is shown in a cut presented herewith. To weigh, the hinged door is swung open, the button placed in the little V-shaped pan at the right and the door closed to keep



New precision balance for assay work

off drafts. The index lever is rotated until the balance pointer comes to zero. Then the index pointer indicates the weight of the sample, the value being read directly in milligrams on the dial. The beam is damped, so there is hardly any overswing. Below the V pan a funnel and tube are located in such a way that the button, after it is weighed, can be shoved off the pan, dropping through the funnel and out the end of the tube shown below the base of the balance. Accuracy is determined by the range of balance selected, this in turn being determined by the weight of the largest sample to be measured. For a range of 0-5 mg., the value per scale division is 0.02 mg. and the vernier reading possible 0.002 mg. For a range of 0-2,000 mg. these readings are, respectively, 10 mg. and 1 mg.

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BULLETINS

Pyrometer—Bacharach Industrial Instrument Co., 7000 Bennett St., Pittsburgh, Pa. Bulletin No. 296; Holborn-Kurlbaum optical pyrometer.

First Aid — E. D. Bullard Co., 275 Eighth St., San Francisco, Calif. Bulletin "Unit First-Aid Kits for Every Requirement."

Gas Pumps—Roots-Connersville-Wilbraham division of International Stacey Corporation, Connersville, Ind. Bulletin 31-B10, "R-C-W Victor Acme Gas Pumps."

Electrical Controlling Apparatus—Allen Bradley Co., Milwaukee, Wis. Fourteen new bulletins and price sheets for revising the company's loose-leaf catalog.

Scrubber — The Hardinge Co., York, Pa. Bulletin 37, "Hardinge Conical Scrubber."

Magnetic Separators—Magnetic Manufacturing Co., Milwaukee, Wis. Bulletin 20, "Stearns Type Q Magnetic Separators."

Motors — Wagner Electric Corporation, St. Louis, Mo. Bulletin 174, "Squirrel-Cage Motors."

Dredging Pumps — Morris Machine Works, Baldwinsville, N. Y. Bulletin 146, "Morris Type F Heavy-Duty Dredging Pumps."

NEWS OF THE INDUSTRY

WASHINGTON, D. C.

Copper Tariff Passed. Acceptance of the tariff items—on copper, oil, coal—in the tax bill by the Senate is expected to have a steadying effect. Moreover, it takes the tariff out of this year's political campaign. In the debate over the copper tax doubt was expressed as to the advisability of levying a tariff at this time, and many questions were raised as to the soundness of using a revenue bill for tariff purposes. Its psychological effect is expected to be of more value than actual protection. Perhaps the chief effect of the action of Congress is to put an end to tariff antagonism between the two parties. . . . The Senate has refused to broaden the tariff provisions of the revenue bill to add an import tax on phosphate rock. . . . Congress is unlikely to change the requirements for assessment work on mining claims. Demand is evident from interests holding mining claims, for a suspension of assessment requirements, but opposition to change is strong. . . . Withdrawals of foreign deposits cannot force the United States off the gold standard. Treasury officials would be relieved if the French were to withdraw their deposits, so that the American money market would not be affected constantly by French politics. The United States has more gold per dollar of bank credit than at any time except during the unusual peaks in certain recent years. The present gold supply is ample for a large expansion of business.

Modification of Anti-Trust Laws. Voluntary agreements among sellers to avoid destructive competition is necessary for the economic welfare of the nation, according to the National Association of Manufacturers. Congress is to be petitioned to pass an "Emergency Industries Preservation Act" to pave the way for modification of anti-trust laws.

The anti-trust law plank of the manufacturers' organization, adopted recently, follows:

"We endorse the operation of the Anti-Trust Laws in respect to their intended purposes—the prohibition in private industry of monopoly, oppression of competitors tending thereto, and unfair trade practices. But experience has clearly shown that prohibition of cooperative agreements between sellers (by judicial interpretation of the Sherman Act) has denied to producers equality of privilege with consumers in determining economic levels of price and production, and has thereby fostered widespread industrial and social maladjustment. Since the producing functions of labor and capital must supply the ability to consume, it follows that protection of payrolls and investments is the imperative condition precedent to the preservation of mass purchasing power, to the permanent supply of consumer needs, and to the maintenance of the essential source of all taxing power. It is therefore the primary necessity of a nation's economic welfare.

"For these reasons, we urge:

"(1) That Congress immediately investigate the workings of all phases of our anti-trust laws and, without sacrifice of their above stated fundamental purposes, enact such legislation as will permit voluntary agreements between sellers to such an extent as may be necessary:

(*a*) To avoid destructive competition.
(*b*) To avoid wastage of materials.
(*c*) To avoid the constant incentive to capital concentration otherwise inevitable.
(*d*) To help stabilize payrolls.
(*e*) To preserve earnings—the fountain source of all tax revenues.
(*f*) To restore industrial equilibrium by restoring equality of privilege between buyer and seller.
(*g*) To promote the public interest by giving balanced weight to the needs of capital and labor at least equally with that of the consumer.

"(2) That, pending this determination of the proper form of permanent legislation Congress immediately pass an "Emergency Industries Preservation Act" conveying this necessary relief temporarily under the supervision of the Federal Trade Commission."

ARIZONA

Arizona Coppers Curtail. Magma has announced suspension of nearly all operations during June, development crews alone to be employed during the summer months. The company plans to provide as much work as possible for senior employees. . . . United Verde Extension put into effect a reduction of salaries and wages of about 10 per cent on May 1. . . . On May 7 Inspiration Consolidated announced suspension of operations. . . . Shutdown of the plant of Miami Copper became effective on May 16.

Activity in Gold and Vanadium. S. Houghton is building a 200-ton cyanide plant near Mammoth, Pinal County, Ariz., at a cost of $100,000, to handle ores being developed in the district and now being trucked to the A.S.&R. plant at Hayden, Ariz. . . . According to Gaetano Naddeo, president, United Vanadium will begin operations at its property in Dripping Springs Wash, Gila County, early in June, when $125,000 will be available for operation of mine and mill. Herman Hertz, general manager, will be in charge.

UTAH

A.S.&R. Blows in Furnace. One blast furnace at its lead smelter near Murray was blown in by American Smelting & Refining on May 20. About 225 men were given employment.

Tintic District Operations. Work of International Smelting is confined to the running of three headings from the North Lily shaft near Dividend, 3 miles from Eureka, Utah, into the Tintic Bullion property, northwest of the shaft, which was recently acquired from Chief Consolidated. One crosscut is being advanced from the 1,200 level in Tintic Bullion ground and two crosscuts are being advanced from the 900 level.

Utah Copper Suspends. All operations will be suspended by Utah Copper from June 15 to July 18.

CALIFORNIA

Placer Mining Revival. Construction of flumes and pipe lines at its hydraulic property in Trinity County is announced by La Grange Placer . . . A ten-stamp mill and flotation equipment have been installed at the Shasta-Trinity property near Hayford, Trinity County. . . . Near Etna, Siskiyou County, three giants are in operation on the Parker & Smith hydraulic property. . . . Mountain Laurel and Wike claims near Sawyer's Bar have been sold to A. R. Patterson. The White Bear property, near Salmon Mountain, is being reopened. . . . Peters mine, in Indiana Valley, Plumas County, has been reopened, and a small mill installed. Adjoining the city limits of Oroville, E. I. West and associates will work a placer area in the bed of the Feather River. . . . Graywing Extension Mining is drilling test bores near Folsom and proving a portion of a Tertiary river channel. Mormon Hill lode claims near Georgetown have been leased by R. G. Dent, and a milling plant is to be erected. Calmo drift property near Angels' Camp is to be reopened as the Mound City gold mine, by P. Alexander. Equipment is being installed at Loope, Alpine County, on the claim of Colossus Mining Corporation. . . . Southern California Gold Mines, Ord Mountain district, northeast of San Bernardino, has completed erection of equipment to output 500 tons daily of gold-copper ore. The company plans to erect a mill.

Copper, Gold, and Silver Discussed. On May 18, at a conference on mining in San Francisco, convened by the Chamber of Commerce of the United States, John Hays Hammond stated that economists are beginning to realize that silver is needed for the business of the world. Horval A. Smith, of Warren, Ariz., advocating a copper tariff, said: "The copper tariff pathway was marked 85 years ago, and in 1869 its way was blazed and maintained for more than two decades, during which time was founded the great copper-mining industries of the West that now produce 90 per cent of our domestic copper. The copper-mining industry was initiated, maintained, and brought to a self-sustaining basis through the medium of adequate tariff protection." . . . J. W. Finch, of Moscow, Idaho, maintained that "The future of gold is predicated on its continued use as money. If the gold standard is abandoned, gold mining will stop and, I believe, forever. If gold mines are closed down for any considerable period they will never be reopened." Urging need for new gold discoveries, W. Mont Ferry, of Salt Lake City, prophesied that the United States will adopt a measure which recog-

The editorial representative of *Engineering and Mining Journal* in Johannesburg, Owen Letcher, contemplates the No. 5 shaft of Crown Mines, the second largest gold producer in the world, which maintained its regular annual dividend payments of £1,260,000 in 1931

nizes silver as basic money metal. Such a measure, he said, will deflate the dollar value and thereby raise commodity prices; it will strengthen and broaden the gold basis of credit by adding to the metallic base an amount of silver against which real and not credit currency may be issued. No fixation of silver price should be attempted. Experience has taught the futility of attempted price fixation. The speaker quoted statistics to show the insufficiency of the eleven billion dollar gold base to support a credit structure estimated as high as $500,000,000,000.

Accident Compensation. Arthur B. Foote has investigated the subject of mining accidents in California and finds that the rate is a little above the average, but not nearly as much as the cost of compensation is above that of compensation insurance in other states. The California insurance rate has doubled since 1923, but accidents have not increased. He places much of the responsibility on the Compensation Commission and would so change the compensation law as to reduce the insurance "loading" of private insurance carriers from 40.6 per cent to 15 per cent, and he would also place a limit on the amount that an accident to one person may cost. Opinion is growing in California that the Accident Compensation law should be repealed, and a new and greatly simplified law should be substituted, which should provide for a non-partisan, non-political administration, a definite limitation on compensation for total and partial disability, and a clear definition of employer and employee responsibility. When and where compensation should begin, the measure of compensation, and maximum limits of compensation—these matters should not be left to the discretion of a politically appointed commissioner.

MONTANA

Gold Mining Receives Attention. Northern Pacific is exploiting a travertine deposit near Gardiner, on the Yellowstone Park branch of the railroad. The rock resembles the deposits in Italy, and will be used for building purposes. . . . Big Chief Syndicate has taken over and will operate the Big and Little Chief claims, in White Horse Gulch, Broadwater County, under the supervision of L. S. Ropes. . . . Big Belt Gold, operating a deep placer on Magpie Gulch, near Canyon Ferry, is changing its power from gasoline to electricity, and connecting with the line of Montana Power. It is installing pump, hoist, and gold washers. This is the only deep placer working in Montana. . . . Prospectors and placer miners are moving their families into tents and cabins along the streams of western Montana, prospecting and recovering gold. This work appeals especially to the unemployed, many of whom are thus making a living. . . . Consolidated Mining & Milling has been incorporated in Montana. Operations are being conducted at the Ida mine, 18 miles southeast of Boulder, Jefferson County. . . . Vigilante Mining Association held a meeting at Pony, Madison County, on April 30, at which was formulated a plan to ask Congress or some other government agency, such as the Reconstruction Finance Corporation, for a revolving fund of $30,000,000 to stimulate gold mining and production on a profit sharing basis. The fund is to be divided into three parcels of $10,000,000 each to be available to the north, south, and central sections of the territory outlined—west of a line drawn north and south through Denver and the Black Hills. The committee states that "The plan, if successful, will put many men to work, and will take care of some who now are dependent on local relief associations, and at little additional cost. This will bring back morale and will go far to allay social unrest. By stimulating gold production, almost all other lines of endeavor will be stimulated, and so will help to end the depression."

PACIFIC NORTHWEST

Washington and Idaho Developments. Sunshine Mining is deepening its main shaft on Big Creek, near Kellogg, Idaho, and continuing to develop the property laterally. . . . Satisfactory results attend diamond drilling exploration of Century Zinc on the property of Metaline Mining & Leasing, near Metaline Falls, north of Spokane. This drilling may take three months. On the results depends the exercise of an option for control of the Metaline company. . . . According to a statement issued by the company, the Morning and Page mines of Federal Mining & Smelting, in the Coeur d'Alenes, Idaho, recently closed, may be reopened in October if conditions warrant.

Students as Placer Miners. According to A. E. Drucker, dean, thirty students of the college of mining and geology of the Washington State College, Pullman, Wash., have joined the staff of a placer-mining company operating in Washington, and will spend the summer in active field work. Instruction in panning and gold recovery has been a part of the school curriculum for some time past, and an appreciable augmentation in the output of gold from the rivers of western Montana, eastern Washington, eastern Oregon, and Idaho is expected to result from the decision of several hundred students to become prospectors.

Profits From Low-Grade Gold Ore. Alaska Juneau in April mined and trammed to mill 326,240 tons. Estimated results were: operating revenue, $263,000, or 80.61c. per ton; operating expenditures, $163,000, or 49.96c. per ton; operating profit was $100,000 or 30.65c. per ton. Interest received, less outside prospecting, brought the surplus to $101,800 for the month. . . . Continental Gold is taking active steps to put the placer properties at Kougorok, Alaska, in operation soon. Extensive equipment was recently shipped from Seattle to Davis Landing.

New Map. A map of the Metaline mining district, in Washington, is being completed by Charles H. Goodsell, U. S. Mineral Surveyor, Spokane. It includes all claims throughout this country and embraces those groups of the same ore characteristics on the British Columbia side.

Claims Acquired. The Jumbo claims, 364 acres, on Lost Creek, near Salmo, B. C., have been acquired by residents of California. The properties include the Queen, Motherlode, Nugget, and Reno mines, the ores of which are principally gold and silver.

Yankee Girl Ships. Yankee Girl, at Ymir, B. C., is now first among the custom contributors to the smelter of Consolidated Mining & Smelting of Canada. It is producing about 1,000 tons of ore per month.

MICHIGAN COPPER COUNTRY

Copper Companies' Taxes Reduced. The Michigan State Tax Commission has reduced the assessed valuation of the mines of Houghton and Keweenaw counties from $28,562,661 to $19,992,470. Valuation of Houghton County mines was fixed at $14,132,470, compared with $20,347,661 in 1931, and that of Keweenaw County at $5,860,000, against $8,215,000 last year. The assessed valuation of the Calumet & Hecla properties in Houghton County has been reduced from $14,650,000 to $10,580,000 and in Keweenaw County from $5,575,000 to $4,040,000. Copper Range has been reduced from $2,765,000 to $1,900,000; Isle Royale from $950,000 to $645,000; Quincy from $900,000 to $435,000; Arcadian from $50,000 to $25,000; La Salle from $80,000 to $25,000; Superior from $20,000 to $10,000; Mohawk from $1,850,000 to $1,380,000; Seneca from $700,000 to $350,000; and Douglass Copper from $40,000 to $30,000. Refined copper in Houghton County is valued at $542,470 and in Keweenaw County at $60,000. The Franklin, Hancock, and Winona properties are returned to the local assessor for valuation. In his recommendations to the commission, Frank G. Pardee, state appraiser of mines, stated that under the prevailing conditions the computations of valuations cannot be measured by past records of the properties. World oversupply and overdevelopment in copper have brought entirely new factors into play. For these reasons, the valuations this year are not intended to represent the sales values of these mines, but as a measure of the relative worth of the properties among themselves. Mr. Pardee says the mining companies, in making every effort to keep their properties working in the face of extremely discouraging conditions, have found it necessary to sacrifice rich ground in order to lower the cost per pound and in most cases practically to cease development work.

Record Low Production Costs. Mohawk costs are the lowest in the company's history. No. 6, the only operating shaft, is permanently bottomed at the 32 level and the lode depth is rich in copper. Developments and construction work are no longer necessary, and wage reductions likewise have contributed to favorable costs. The operating cost, including mining and stamping, appear to be under 5c. per pound. Mohawk is not smelting any of its mineral, awaiting a better market for the metal, but total costs probably will not exceed 6c. Copper and mineral stocks on hand are estimated at 34,000,000 lb. of refined copper.

IRON COUNTRY

Improvement in Slusher Hoisting. An automatic double-drum electric slusher hoist has been developed by Ingersoll-Rand and General Electric companies, and tested out at the Utica mine of Pickands Mather on ore transfer underground. Operation is as smooth as if under manual control. The hoist started in March and operated until the mine closed down in May. The sequence of operations is as follows: When the miner in the upper levels is ready to pull dirt he pushes a button that starts the transfer hoist perhaps 50 ft. below, controlled through contacts on the pull-rope drum. A scraper is pulled through the ore pile at 50 ft. per minute, increases its speed to 450 ft., slows down when near the raise to 50 ft., stops for an instant near the hoist, reverses and returns to the ore pile at 550 ft. per minute. The length of haul is over 600 ft., and power is supplied by a 35 hp. motor.

Mesabi Chief Starts Production. Operations started at the Mesabi Chief mine of the M. A. Hanna Company on May 24. This is the first open-pit mine to start ore production this year. A five-day week schedule is in force.

Ore Movement Light This Year. Only a limited amount of labor will be employed at the mines next winter. To anticipate such conditions, all the range welfare clubs, as well as the mining companies, are encouraging the men to grow large gardens during the summer that will carry them through with ample food for the coming winter. Available ground around the towns is being turned over to committees, who see to its plowing and preparation into garden plots.

Oliver Retires Old-Timers. Two well-known mining men of the Mesabi range were recently retired on pension by the Oliver Iron Mining Company. "Hank" Downing, of Coleraine, who has been mine superintendent in that district for the last 25 years, and "Leo" Mitchell, mine superintendent at Chisholm, have reached the "Age of Leisure." Fitting farewell banquets were tendered them.

NEVADA

Interest in Gold Properties. Anaconda Copper has taken over on bond and lease the Mountain View claims, northwest of Gerlack, Nev., located about three years ago by W. S. Bretz and John Durkin. This is the third important deal Mr. Bretz has turned in Nevada. The first was the Opalite cinnabar property on the border line of Nevada and Oregon, taken up by the Bradleys, of San Francisco, under the name of Mercury Mines. . . Montezuma Gold Mines has taken over the claims near Silver City, on the Comstock Lode, formerly owned by M. J. King and a Mr. Blair. Montezuma Gold is headed by John W. Ross. Money has been raised for purchase of claims, opening of old workings, equipping a one-compartment and manway shaft, and purchase of air compressor and drills. This shaft will develop the orebody first opened by King and Blair. . . . Operators of the Chollar, Gould & Curry and Savage mines on the Comstock plan to explore virgin ground south and west from the Overman shaft, by crosscut. . . . Rosetta Mining has put a force of miners to work to explore the north drift, to determine extent of the orebody in that direction. . . . E. W. Geier and Colin Timmons have taken over the Ellendale mine, about 40 miles east of Tonopah. Their first work will be the building of a good road to the mine, and to provide a water supply. . . . A.S.&R. engineers recently visited the Rio Tinto mine, 90 miles from Elko. The Rio Tinto has developed a remarkable deposit of high-grade copper ore, which will pay to ship even at low prices. The gold and silver content is low, but copper runs 40 per cent and over. From 500 to 700 tons of high grade is on the dumps, about 100 tons of which is to be shipped to the A.S.&R. smelter. The trucks will go back from Elko loaded with supplies and railroad ties from which cabins will be constructed for miners. . . . Senator Edgar S. Hurley, of Alameda County, Calif., is building a 35-ton custom mill, for the Gilbert district, consisting of two 15-hp. engines, a crusher, grinding mill, two concentrating tables and two amalgamators. . . . Consolidated Coppermines suit against Nevada Consolidated was temporarily concluded on May 20. More arguments will be heard in the fall. Each company has employed a small army of attorneys.

The Golden Mile, Kalgoorlie, Western Australia, where labor employment figures are rising steadily, and where a considerable increase in gold output is expected for 1932

AUSTRALASIA

Wiluna Increases Reserves. The consulting engineer to Wiluna Gold, Western Australia, H. E. Vail, states that the original mine estimates will be realized. Discovery of the new western orebody on the 625-ft. level, where a borehole encountered ore 51 ft. wide valued at 44s. 6d., was important. This orebody supplements the main lodes, on the possibilities of which the company had been floated, the plant constructed, and development work carried out. Increase in ore reserves by an average of 65,000 tons per month is reported, with progress in metallurgical extraction. . . . Broken Hill Proprietary, which has an option over the Golden Zone at the north end of Kalgoorlie, has made a trial crushing at the Boulder Perseverance mill, with satisfactory results. A diamond drilling plant is being erected at the old Rothesay mine, 45 miles east of Perenjori, over which the company also has an option. . . . Six months' exemption on leases at Coolgardie, Edjudina, and Cue has been granted to H. Mandelstamm, now in London to secure £500,000 for each of the leases, which will be developed as low-grade properties. Mr. Mandelstamm expects to return before June, to commence boring operations to test values and determine tonnage. . . . Discovery of gold is reported from the Lady Mary mine, at Sandstone. A company may be floated to develop the property. A crushing of 20 tons yielded 240 oz.

Uranium-Radium Developments. Australian Radium Corporation, developing low-grade radium deposits at Mount Painter, South Australia, is meeting with encouraging results. Autunite and torbernite have been found in surface workings, but little information is available as to the nature of the ore beneath the oxidized zone. A concentration method has been adopted, and the concentrate assays 8 per cent uranium oxide. The company hopes to raise the grade of concentrate, and erect a new plant at Melbourne, Victoria. . . . Output of gold from Bendigo, Victoria, continues to gain. The principal mines are planning to increase tonnages milled, and gold yield. A new company has been formed to unwater and equip the South Blue Consolidated, the shaft of which is to be deepened. . . . With a capital of £30,000, Sunstar Gold Mining has been formed to develop the Sunstar mine, at Daylesford, Victoria. . . . Mount Lyell, Tasmania, for the quarter ended March 31, treated 81,936 tons, producing therefrom 11,549 tons of concentrate. Gold and silver, in the anode mud of the company's electrolytic refinery, is dispatched in monthly parcels to a plant at Port Kembla, N.S.W. Electrolytic copper is also shipped to this plant for fabrication, the products being taken by Metal Manufacturers, a copper and brass rolling and cable manufacturing company, in which Mount Lyell holds a substantial interest.

Electrolytic Zinc Not Curtailing. Electrolytic Zinc announced in December, 1931, that production was to be curtailed, but the reduction has been insignificant. For the four weeks ended April 6, production of electrolytic zinc was 4,000 tons; silver-lead residue contained 145 tons lead and 13,000 oz. silver. . . . Sulphide Corporation's Central mine and flotation mill at Broken Hill, N.S.W., recommenced operations in April after having been idle since December, 1930. The company is employing 480 men. For March, crude ore treated by Zinc Corporation, Broken Hill South, and North Broken Hill totaled 76,278 tons, producing 14,869 tons 74 per cent lead concentrate, and 13,140 tons of 52.5 per cent zinc concentrate. Output of zinc concentrate at Broken Hill just meets the requirements of Electrolytic Zinc, and no surplus is available for export.

North Broken Hill's new shaft has reached a depth of 1,301 ft., and is in country rock. Following a dispute with the contract miners, an agreement has been reached whereby, if the wet-bulb temperature in the shaft exceeds 76 deg. F., a six-hour shift is to be worked.

Mount Isa Adds to Plant. During March, Mount Isa treated 45,643 tons, recovering 9,034 tons of 43.8 per cent lead concentrate. Recovery of lead was 80 per cent. The first of three new sintering machines has been installed, and tenders have been let for the erection of No. 3 blast furnace, specially designed to treat carbonate ore. The Queensland Government is proceeding with the construction of coke ovens at Bowen; the coke produced will be despatched to Mount Isa. . . . The Queensland Government smelter at Chillagoe has sufficient copper ore on hand for twelve months' operation. Considerable cupriferous gold ore and quartz gold ore are being purchased, in addition to a dump of 10,000 tons of flue dust, assaying 8 per cent copper and 1 dwt. gold per ton, from Mount Elliot Mines.

Gold Mining Stimulates Employment. At a meeting of the Western Australia Chamber of Mines, the number of men employed in the gold-mining industry throughout the State was given as 6,291, compared with 4,284 a year before. Of that number 2,342 were employed on the Golden Mile, compared with 1,996 in 1930. . . . Gold output of Western Australia for 1932 will probably exceed the 1931 output of 510,000 oz. Abundant low-grade ore is available. Ore reserves of the principal Kalgoorlie mines exceed 2,100,000 tons assaying 8 to 9 dwt. Development work now being carried out by Lake View & Star, North Kalgoorlie, and Great Boulder & Boulder Perseverance is steadily adding to reserves.

Dredging Results. Bulolo Gold has announced that during the 28 days to April 25, 243,700 cu.yd. was treated for a yield of $57,000, or over 23c. per yard, compared with 21c. indicated by boreholes. The capacity of the dredge is in excess of the estimates, and the other dredges will be completed as quickly as possible. . . . Mining Trust reports that a satisfactory trial run has been made with the power plant of New Guinea Goldfields and that the transmission lines will be completed soon.

American Interests in Australia. American Smelting & Refining has acquired an option over the Shackell gold mine near Grove Hill, in Northern Australia, said to be low grade. . . . Option of purchase over Mararoa Gold Mining leases at Reidys, W.A., has been granted to United Verde, of New York. Two company representatives are examining the property. The option is for 2½ years, the terms are £50,000 cash; or, if a company be formed, a 25 per cent fully paid interest in such company.

Mining in New Caledonia. According to the *Bulletin du Commerce*, the monthly production of Société du Nickel, operating in New Caledonia, amounts to about 5,534 tons of nickel matte, exported to France and Belgium. Shipments of chrome ore from the island in 1931 amounted to about 38,000 tons, most of which went to the United States. The chrome mining companies —principally Société Caledonia, Société Chrome Chimique, and Société Tiebaghi —are maintaining output despite the depression elsewhere.

BRITISH SOUTH AFRICA

Wit Deep Strikes Bonanza.—A strike of visible gold was made on April 18 on the 17 intermediate level of the Witwatersrand Deep. Specimens contain from 60 to 80 per cent gold—the richest discovery in any Witwatersrand mine. The find confirms in some respects the theory that at one time a great river ran from the north of the Witwatersrand, through a delta into a lake, or into some large water surface. The river at times became torrential, and carried with it large pebbles and gold in suspension. These pebbles and gold particles ultimately formed part of the sediment deposited in the delta or lake bed. The gold had been derived from denudation of gold-bearing rocks, since vanished. At its mouth the gold-bearing river probably ran through the delta by way of several effluents, and it is said that the richest zones of reef in the Rand, when picked out, almost define the zones of reef of these effluents. The theory that the Wit Deep pocket was the result of heavy precipitation through some chemical agent in the dike is in conformity with existing theories regarding the origin of the reef. The occurrence of similar rich segregations of gold in quartz has been noted in many parts of the Rand, and these auriferous quartz bodies have been used to support the hydrothermal theory of the origin of gold, especially by Prof. L. C. Gratton. . . . An unusually heavy fall of rock occurred at the Geldenhuis Deep on April 17, shaft No. 3 of the Jumpers' Deep section. Tonnage and profits for May will be seriously affected. No casualties were reported, but the tremor was felt over a wide area.

Rhodesian Coppers. In April, Roan Antelope treated 84,300 short tons averaging 3.79 per cent copper, and produced 2,113 long tons of 99.57 per cent blister. . . . Rhokana Corporation is producing, but not reporting production. Results obtained are better than was expected. Concentrate is being smelted direct without fluxes, and with satisfactory results. . . . Efforts are centered in Northern Rhodesia on keeping the costs down while producing at

only a fraction of rated capacity. With reduction of working force has come an increase in the efficiency of labor. But the cost of most supplies and of transportation is high, as compared with costs in America, and no further reduction of freight rates can be hoped for as long as production is on a reduced scale. . . . Results of two more boreholes are reported from Kansanshi. One of these penetrated 31 ft. of 5.5 per cent ore; the other, 40 ft. of 3.2 per cent; and then, deeper down, 27 ft. of 7.1 per cent ore.

Resists Currency Depreciation. Union of South Africa has decided to remain on the gold standard, according to the verdict of a committee appointed by parliament last February. The alternative was to follow the example of other units of the British Empire in South Africa and adopt sterling as standard. Exporters in the Union are at a disadvantage, of course; but the government is prepared to protect them by subsidy. If the Union persists in its plan to issue a new gold coin, to be known as the rand, a blow will be dealt at the proposal made elsewhere to introduce a standard Empire currency.

Lomah Restarts. Lonely Reef Pays Dividend. Authorized capital of the reconstructed Lomah (Rhodesia) Gold Mines is £150,000. Sufficient working capital has been provided and mining operations have been restarted. Lonely Reef declared its usual 10 per cent interim dividend for 1932.

WEST AFRICA

Gold Mines Active. Output and grade of Taquah & Abosso are likely to be adversely affected during the next few months, until new compressors and other equipment, now being shipped, are installed. . . . Ashanti Goldfields profits for seven months of the financial year amount to £389,446, including premium equivalent to £667,620 per annum, or 133.5 per cent on the present capital. Without premium, profits equal 86 per cent. An interim dividend of 25 per cent has been declared.

BELGIAN CONGO

Gold-Tin Developments in East Congo. Kilo-Moto produced 1,924 kilos gold in the first quarter of 1932. Raymond Anthoine, consulting engineer, recently returned to Belgium from an inspection trip, reports organized activity in planning for production and transportation of supplies and products—gold and tin—from the Oriental Province of the Belgian Congo. The companies operating, all Belgian, include Soc. des Mines d'Or de Kilo-Moto; Soc. Minière de Surongo; Soc. Minière des Grandes Lacs Africains; Minière de la Télé; Minière de l'Aruwimi-Ituri; Forminière; Comité National du Kivu; Soc. Minière du Bafwaboli Syndicat Minier Africain; Minière de Kindu; Sindicat Minier du Lualaba; Belgikaor; Minière du Maniéma, all with headquarters in Brussels. The Urégа country has been closed for recruiting for other agencies. Alluvial tin has been found and developed in such quantities, according to Mr. Anthoine, that profitable operation of the placers is possible even at present low tin prices. . . . Gold has already been dispatched from the Province Oriental to Matadi, at the Congo mouth, by airplane. This rapid, safe, and economical method will be more extensively adopted in the future. Not only in regard to the product, but in connection with the transportation of staff, government officials and doctors, the delivery to mining and prospecting camps of medicines and general requirements of light-weight character, the airplane is expected to facilitate the opening up of mineralized areas of the Belgian Congo.

YUGOSLAVIA

Trepca's Profits Justify Expansion. Trepca Mines milled 29,354 metric tons of ore in April. Production was 3,391 metric tons of concentrate assaying 78.1 per cent lead and 27.5 oz. silver per ton; and 4,659 metric tons of concentrate assaying 50.3 per cent zinc. During the first quarter of the year 82,151 metric tons was milled. Concentrates totaled 10,845 metric tons, averaging 77.57 per cent lead and 27.9 oz. silver; and 11,725 tons averaging 49.99 per cent zinc. Recovery averaged 92.29 per cent of the lead, and 83.81 per cent of the zinc. Despite low prices, estimated profit for the quarter was £21,291 after all costs, including London expenses, had been met. Development on the 730 meter level cut a further 53 meters of ore, and this main drift and short crosscuts from it showed an average of 9.49 per cent lead and 10.46 per cent zinc. Capital expenditure during the quarter came to £30,389. This was on the addition to the concentrator, so well advanced that it will be brought into commission gradually during the present quarter.

ITALY

Bauxite Deposits Discovered. According to Consul Rollin R. Winslow, Trieste, confirmation is available of reports from Gorizia of discovery of bauxite deposits. The deposit is said to contain from 57 to 66 per cent bauxite, with an average of 63 per cent, higher than the percentage in the Istrian deposits. Experts are said to have estimated for the lower strata as much as 70 per cent bauxite, one of the highest in the world.

BOLIVIA

Interest in Gold. The Huanuni dredge was recently restarted. Patiño Mines contemplates a further reduction in production, to bring the total Bolivian tin output within the limits set by the recent decision of the Tin Producers' Association. The company recently took over the administration of the Araca company, following a consolidation. A similar consolidation with the Oploca company may follow. . . . The Bolivian Government has dropped the exchange value of the boliviano from about thirteen to fifteen and later to eighteen to the pound. Nevertheless, the condition of many of the tin mines is serious. . . . The Tipuani company has extended a development option to Guggenheim interests, operating through the Aramayo organization; drilling will begin soon. Many consider that reasonably good chances exist for important gold developments in northern Bolivia and southern Peru.

BRITISH COLUMBIA

Pioneer Strikes Rich Ore. Pioneer is extracting ore from a new vein, 5 ft. wide. Put through the mill it has shown an average gold content of $10 per ton. The discovery is regarded as highly important. . . . Cariboo Gold-Quartz, operating near Barkerville, in the Cariboo district, contemplates the erection of a 25-ton pilot mill on the property in the immediate future. . . . U. S. Smelting & Refining has not exercised its option on placer properties in the Cedar Creek area of the Cariboo district, the result of tests having been disappointing. . . . Plans for the further development of the Dawson and Aurum gold properties are being made by a Vancouver syndicate, which recently acquired control. . . . French Creek Development has resumed hydraulicking operations on its placer property in the Big Bend district. . . . Wellington-Beaverdale syndicate, operating the Wellington mine, on Wallace Mountain, in the Kettle River valley, has decided to install additional machinery, including a new power plant. In spite of the low price of metals, the Wellington and Bell mines are able to produce high-grade silver-lead ore at a profit. The Carmi mine, in the same locality, after being abandoned for several years, was recently reopened.

MEXICO

Activity in Precious and Base Metals. Potosi Mining, operating in Chihuahua, is handling daily about 2,350 tons of silver-lead-zinc ore, and employing 1,200 men. . . . The Kildun company, at Matehuala, San Luis Potosi, is employing 340 men, and the Peñoles company about 400 men at its properties at Avalos, Zacatecas. . . . El Boleo, a French enterprise operating in Lower California, is employing 2,262 men and producing about 850 tons of copper daily. . . . A force of about 3,417 is handling about 2,000 tons of gold-silver ore at the Dos Estrellas properties in Michoacan and at El Oro. . . . A 500-ton mill is being worked to capacity and 823 men are being employed by the Cia. de Inversiones del Oro, in Zacatecas. . . . A new gold vein is reported in La Navidad Anexas mine, Oaxaca. I. S. Bonillas, examining engineer, estimates a production is 100 kilos monthly, but transportation facilities are deficient.

U.S.S.R.

Japanese Smelt Soviet Ore. According to Consul H. B. Hitchcock, at Nagasaki, a vessel of the Soviet mercantile fleet recently landed 1,000 metric tons of zinc ore for the refinery on Hikoshima and 2,500 tons for that at Miike. This cargo is the first of 10,000 tons for which Mitsui Mining contracted last year with the Soviet trading agency. The source is a mine in the Maritime Province of Siberia, about 180 miles inland from Vladivostok. The mine had been opened in Czarist times and was the chief source of supply for the Hikoshima refinery, then an enterprise of the Suzuki Shoten, from which the Mitsui interests later acquired it. Imports of ore ceased entirely after the Russian revolution, but the mine has now been reopened by the Soviets.

Daily and Average Monthly Prices of Metals

May, 1932

United States Market

May	Electrolytic Copper, Refinery	Straits Tin New York	Lead — New York	Lead — St. Louis	Zinc St. Louis
2	5.525	19.875	3.00	2.90	2.55
3	5.525	20.450	3.00	2.90	2.50@2.55
4	5.525	22.000	3.00	2.90	2.50@2.55
5	5.40@5.525	22.00@22.20	3.00	2.90	2.50
6	5.425	22.050	3.00	2.90	2.50
7	5.40	22.000	3.00	2.90	2.50
9	5.275@5.40	21.00@21.15	3.00	2.90	2.50
10	5.275@5.30	20.625	3.00	2.90	2.475@2.50
11	5.275	21.375	3.00	2.90	2.40
12	5.150@5.275	20.750	3.00	2.90	2.375@2.40
13	5.150@5.275	20.400	3.00	2.90	2.35@2.40
14	5.275	20.400	3.00	2.90	2.35@2.40
16	5.275	20.450	3.00	2.90	2.35@2.40
17	5.150@5.275	20.375	3.00	2.90	2.275@2.35
18	5.150	20.850	3.00	2.90	2.325
19	5.025	21.325	3.00	2.90	2.35@2.40
20	5.025@5.150	21.250	3.00	2.90	2.375
21	5.025@5.150	21.100	3.00	2.90	2.450
23	5.025	20.875	3.00	2.90	2.500
24	5.025@5.150	21.100	3.00	2.90	2.55@2.70
25	5.150	21.000	3.00	2.90	2.875
26	5.025@5.150	20.750	3.00	2.90	2.875
27	5.025@5.150	20.575	3.00	2.90	2.875
28	5.025@5.150	20.550	3.00	2.90	2.875
30		Holiday			
31	5.025@5.150	20.400	3.00	2.90	2.80@2.875
Av. for Month	5.237	20.948	3.000	2.900	2.532
		Averages for Week			
May					
4	5.515	19.992	3.000	2.900	2.554
11	5.365	21.538	3.000	2.900	2.481
18	5.223	20.538	3.000	2.900	2.358
25	5.077	21.108	3.000	2.900	2.533
		Calendar Week Averages			
May					
7	5.477	21.413	3.000	2.900	2.517
14	5.267	20.771	3.000	2.900	2.421
21	5.140	20.892	3.000	2.900	2.369
28	5.088	20.808	3.000	2.900	2.771

Silver, Gold, and Sterling Exchange New York and London

May	Sterling Exchange "Checks"	Sterling Exchange "90-Day" Demand	Silver — New York	Silver — London	Gold, London
2	365.625	366.125	27.250	16.6875	112s 9d
3	365.875	366.625	27.000	16.5625	113s 1d
4	366.750	368.125	27.125	16.6250	112s 6d
5	366.750	368.000	27.250	16.6875	112s 9d
6	368.750	370.000	27.250	16.5625	112s 8d
7	367.000	367.750	27.375	16.6875	.....
9	367.750	369.250	27.625	16.8750	112s 8d
10	367.375	368.875	27.750	17.0000	112s 9d
11	367.250	368.750	27.875	17.0625	112s 11d
12	366.375	367.625	28.000	17.3750	112s 11d
13	365.375	366.500	27.875	17.1250	113s 3d
14	365.000	366.125	28.125	17.1875	.....
16	367.250	368.500	28.750	Holiday	Holiday
17	366.000	367.250	28.500	17.3125	113s 5d
18	367.875	369.625	28.250	17.1250	113s 2d
19	367.625	369.375	27.875	16.8750	112s 11d
20	367.000	368.500	27.750	16.7500	113s 3d
21	367.125	368.625	27.625	16.6250	.....
23	367.375	369.000	27.750	16.6250	112s 9d
24	367.750	369.500	27.750	16.8125	112s 7d
25	368.750	370.750	27.875	16.9375	112s 8d
26	369.250	371.125	27.875	16.8750	112s 6d
27	368.500	371.000	27.625	16.7500	112s 9d
28	370.750	374.750	27.750	16.8125	.....
30	Holiday	Holiday	Holiday	16.8125	112s 6d
31	369.125	373.000	28.000	16.9375	112s 5d
Av. for Month	367.370	.....	27.755	16.868	.....
			Averages for Week		
May					
4	365.708	.....	27.271	.....	.....
11	367.479	.....	27.521	.....	.....
18	366.313	.....	28.250	.....	.....
25	367.604	.....	27.771	.....	.....

Calendar week averages, New York Silver: May 7th, 27.208c.; 14th, 27.875c.; 21st, 28.125c.; 28th, 27.771c.

London Market

May	Copper — Standard — Spot	Copper — Standard — 3 Mo.	Copper — Electrolytic (Bid)	Tin — Spot	Tin — 3 Mo.	Lead — Spot	Lead — 3 Mo.	Zinc — Spot	Zinc — 3 Mo.
2	29.7500	29.6250	34.25	114.375	116.500	11.4375	11.6250	12.2500	12.5000
3	30.3750	30.1250	34.25	120.625	122.750	11.7500	12.0000	12.7500	13.0000
4	30.1875	30.0000	34.00	126.000	128.250	11.6875	11.9375	13.0000	13.2500
5	30.2500	30.0000	34.00	129.250	131.500	11.5000	11.7500	13.0000	13.1875
6	29.6250	29.6875	34.00	129.500	131.750	11.3750	11.6250	12.9375	13.1250
9	29.6250	29.8125	34.00	124.500	126.750	11.0625	11.3750	12.8125	13.1250
10	29.3750	29.5000	33.75	120.000	122.000	10.8125	11.1250	12.5000	12.8125
11	29.4375	29.6250	33.75	124.375	126.375	10.9375	11.2500	12.6250	12.9375
12	29.4375	29.5625	33.75	121.000	123.000	10.8125	11.1250	12.3750	12.6875
13	29.0000	29.1875	33.25	120.750	122.500	10.7500	11.1250	12.3750	12.6875
16					Holiday				
17	28.2500	28.4375	32.75	119.000	121.125	10.6875	11.0625	12.4375	12.7500
18	28.2500	28.4375	32.75	121.375	123.625	10.8125	11.1250	12.5000	12.8125
19	28.4375	28.6250	32.00	125.625	127.625	10.6875	11.0625	12.6875	13.0000
20	28.0000	28.1250	32.00	124.750	127.000	10.5000	10.8750	12.7500	12.9375
23	27.5625	27.6875	31.75	122.000	124.000	10.3125	10.8125	12.2500	12.5000
24	27.5000	27.6250	31.75	123.000	125.250	10.2500	10.7500	12.5000	12.6875
25	27.5000	27.5625	31.75	123.375	125.625	9.9375	10.3125	12.1250	12.3750
26	27.4375	27.3750	31.75	121.125	123.250	9.5625	10.0625	12.0000	12.1875
27	27.2500	27.1875	31.50	120.750	123.125	9.7500	10.2500	11.9375	12.1250
30	26.5000	26.4375	31.50	117.750	119.500	9.7500	10.2500	11.8125	12.0000
31	25.7500	25.7500	31.00	118.875	121.000	9.7500	10.2500	11.4375	11.6250
Average for month	28.548	.....	32.833	122.286	.....	10.673	11.036	12.432	12.682

The United States quotations are our appraisal of the major markets for domestic consumption based on sales reported by producers and agencies. They are reduced to the basis of cash, New York or St. Louis, as noted. All prices of domestic class are in cents per pound.

Copper, lead and zinc quotations are based on sales for both prompt and future deliveries; tin quotations are for prompt delivery only.

Quotations for zinc are for ordinary Prime Western brands. Zinc in New York is now quoted at 0.35c. per pound above St. Louis, this being the freight differential. Contract prices for High-Grade zinc delivered in the East and Middle West are 1c. above St. Louis

Silver and Sterling Exchange

	New York		London Spot		Sterling Exchange	
	1931	1932	1931	1932	1931	1932
January	29.423	29.780	13.810	19.623	485.260	342.515
February	26.773	30.136	12.432	19.573	485.551	345.141
March	29.192	29.810	13.524	18.336	485.596	363.463
April	28.279	28.298	13.120	16.923	485.764	374.731
May	27.650	27.755	12.858	16.868	486.188	367.370
June	27.250	.....	12.707	.....	486.291	.....
July	28.255	.....	13.197	.....	485.349	.....
August	27.524	.....	12.815	.....	485.284	.....
September	28.180	.....	14.101	.....	451.245	.....
October	29.538	.....	17.153	.....	388.029	.....
November	32.223	.....	19.393	.....	371.130	.....
December	30.120	.....	20.023	.....	336.798	.....
Year	28.700	.....	14.594	.....	452.707	.....

New York quotations, cents per ounce troy, 999 fine. London, pence per ounce, sterling silver, 925 fine. Sterling exchange in cents.

Copper

	F.O.B. Refinery		London Spot			
	Electrolytic		Standard		Electrolytic	
	1931	1932	1931	1932	1931	1932
January	9.838	7.060	44.938	39.459	47.524	46.200
February	9.724	5.965	45.372	36.917	47.950	41.381
March	9.854	5.763	44.818	33.039	47.699	36.786
April	9.392	5.565	42.694	29.943	45.375	34.190
May	8.665	5.237	38.897	28.548	42.175	32.833
June	8.025	.....	35.827	.....	38.966	.....
July	7.698	.....	34.402	.....	37.293	.....
August	7.292	.....	32.572	.....	35.388	.....
September	6.988	.....	31.503	.....	36.148	.....
October	6.775	.....	34.957	.....	41.000	.....
November	6.558	.....	35.854	.....	41.190	.....
December	6.580	.....	38.273	.....	44.409	.....
Year	8.116	.....	38.342	.....	42.093	.....

New York quotations, cents per pound. London, pounds sterling per long ton.

Lead

	New York		St. Louis		London			
	1931	1932	1931	1932	1931 Spot	1931 3 Mos.	1932 Spot	1932 3 Mos.
January	4.802	3.750	4.604	3.550	13.872	13.905	15.084	15.128
February	4.552	3.712	4.340	3.499	13.444	13.550	14.560	14.571
March	4.527	3.150	4.276	2.993	13.128	13.355	12.345	12.634
April	4.412	3.000	4.164	2.900	12.375	12.606	11.223	11.503
May	3.818	3.000	3.651	2.900	11.491	11.778	10.673	11.036
June	3.917	.....	3.761	.....	11.582	11.952	.....	.....
July	4.400	.....	4.225	.....	12.731	12.899	.....	.....
August	4.400	.....	4.225	.....	11.944	11.994	.....	.....
September	4.400	.....	4.217	.....	11.932	12.026	.....	.....
October	3.964	.....	3.773	.....	13.227	13.270	.....	.....
November	3.937	.....	3.758	.....	14.577	14.491	.....	.....
December	3.792	.....	3.592	.....	15.188	15.361	.....	.....
Year	4.243	.....	4.049	.....	12.958	13.099	.....	.....

New York and St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Tin

	New York		London	
	1931	1932	1931	1932
	Straits		Spot	
January	26.137	21.804	115.798	140.219
February	26.315	22.018	117.919	139.143
March	27.065	21.863	121.852	129.810
April	25.222	19.244	112.775	108.935
May	23.221	20.948	104.331	122.286
June	23.478	.....	104.966	.....
July	24.978	.....	111.478	.....
August	25.738	.....	114.875	.....
September	24.618	.....	117.813	.....
October	22.723	.....	126.932	.....
November	22.779	.....	132.857	.....
December	21.328	.....	138.909	.....
Year	24.467	.....	118.375	.....

New York quotations, cents per pound. London, pounds sterling per long ton.

Zinc

	St. Louis		London			
	1931	1932	1931 Spot	1931 3 Mos.	1932 Spot	1932 3 Mos
January	4.035	3.011	12.747	13.113	14.416	14.834
February	4.012	2.817	12.303	12.694	13.872	14.289
March	4.002	2.787	12.190	12.676	12.616	13.024
April	3.717	2.725	11.353	11.838	11.670	11.958
May	3.306	2.532	10.484	10.875	12.432	12.682
June	3.416	.....	11.270	11.750	.....	.....
July	3.893	.....	12.280	12.802	.....	.....
August	3.917	.....	11.444	12.028	.....	.....
September	3.744	.....	11.571	12.063	.....	.....
October	3.377	.....	12.733	13.216	.....	.....
November	3.209	.....	13.845	14.247	.....	.....
December	3.149	.....	14.361	14.818	.....	.....
Year	3.640	.....	12.215	12.677	.....	.....

St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Cadmium and Aluminum

	Cadmium		Aluminum	
	1931	1932	1931	1932
January	67.115	55.000	23.300	23.300
February	55.000	55.000	23.300	23.300
March	55.000	55.000	23.300	23.300
April	55.000	55.000	23.300	23.300
May	55.000	55.000	23.300	23.300
June	55.000	.....	23.300	.....
July	55.000	.....	23.300	.....
August	55.000	.....	23.300	.....
September	55.000	.....	23.300	.....
October	55.000	.....	23.300	.....
November	55.000	.....	23.300	.....
December	55.000	.....	23.300	.....
Year	56.010	.....	23.300	.....

Aluminum in cents per pound, 99 per cent grade.
Cadmium, cents per pound.

Antimony, Quicksilver, and Platinum

	Antimony (a) New York		Quicksilver (b) New York		Platinum (c) New York	
	1931	1932	1931	1932	1931	1932
January	7.317	5.976	103.000	64.900	36.000	40.000
February	7.069	6.489	100.205	66.304	34.000	40.000
March	7.127	6.188	100.423	72.537	29.538	40.000
April	6.888	5.746	102.077	72.125	26.346	40.000
May	6.524	5.170	101.140	66.380	24.980	39.500
June	6.342	.....	92.058	.....	37.115	.....
July	6.802	.....	85.808	.....	40.000	.....
August	6.596	.....	80.115	.....	40.000	.....
September	6.542	.....	76.300	.....	40.000	.....
October	6.517	.....	72.385	.....	40.000	.....
November	6.679	.....	68.587	.....	40.000	.....
December	6.231	.....	66.115	.....	40.000	.....
Year	6.720	.....	87.351	.....	35.665	.....

(a) Antimony quotations in cents per pound, for ordinary brands. (b) Quicksilver in dollars per flask of 76 lb. (c) Platinum in dollars per ounce troy.

Pig Iron[1]

	Bessemer		Basic		No. 2 Foundry	
	1931	1932	1931	1932	1931	1932
January	17.50	16.00	17.00	15.00	17.00	15.50
February	17.29	15.68	16.79	14.68	16.79	15.18
March	17.00	15.50	16.50	14.50	16.50	15.00
April	17.00	15.50	16.50	14.50	16.50	15.00
May	17.00	15.50	16.38	14.50	16.44	15.00
June	17.00	.....	15.50	.....	16.00	.....
July	17.00	.....	15.50	.....	16.00	.....
August	17.00	.....	15.50	.....	16.00	.....
September	17.00	.....	15.50	.....	16.00	.....
October	16.92	.....	15.42	.....	16.00	.....
November	16.50	.....	15.00	.....	16.00	.....
December	16.25	.....	15.00	.....	15.75	.....
Year	16.96	.....	15.88	.....	16.25	.....

Iron in dollars per long ton. [1]F.o.b. Mahoning and Shenango Valley furnaces; freight to Pittsburgh, $1.76.

prices for the Prime Western grade.

Quotations for lead reflect prices obtained for common lead, and do not include grades on which a premium is asked.

London prices for lead and zinc are the official prices for the morning session of the London Metal Exchange; prices for copper and tin are the official closing buyers' prices. All are in pounds sterling per long ton (2,240 lb.).

New York silver quotations are as reported by Handy & Harman and are in cents per troy ounce of silver, 999 fine. London silver quotations are in pence per troy ounce of bar silver, 925 fine. Sterling prices represent forenoon market demand. Cables command a premium.

Current Statistics of Production and Stocks of Copper, Lead, and Zinc

Data from American Bureau of Metal Statistics, American Zinc Institute, and Metallgesellschaft

All Figures Except Tin Represent Tons of 2,000 Lb.

	1931								1932			
COPPER — North and South America	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Jan.	Feb.	March	April
Production, refined, daily average	3,313	3,276	3,110	2,909	2,890	2,918	2,693	2,780	.....	.....	.....	.....
Domestic shipments	45,265	50,217	43,144	45,816	40,459	51,348	37,436	36,972	.....	.....	.....	.....
Foreign shipments	26,684	33,251	26,321	29,016	22,124	21,030	20,257	26,032	.....	.....	.....	.....
Stocks, blister and refined	589,345	600,827	620,075	631,880	658,321	.....	.....	722,014	.....	.....	.....	.....
LEAD — United States												
Production, refined, from domestic ore	39,519	30,708	32,157	34,144	31,966	36,546	31,671	33,576	32,180	28,081	30,345	23,236
Production, secondary and foreign	3,598	3,683	4,409	5,453	3,934	4,722	3,820	4,031	4,559	3,920	3,989	3,134
Production, total, daily rate	1,390	1,146	1,179	1,151	1,163	1,331	1,180	1,213	1,185	1,103	1,107	879
Shipments reported	34,081	37,054	42,219	38,590	38,059	34,276	31,216	30,297	27,867	26,319	31,162	26,081
Stocks, end of month	142,370	139,698	133,958	134,977	132,804	139,796	144,057	151,380	160,257	165,933	169,091	169,091
ZINC — United States												
Production, daily average	971	783	689	692	712	699	684	709	723	742	726	688
Domestic shipments	25,831	27,604	28,460	23,599	20,902	21,163	20,327	23,005	22,472	21,896	22,576	18,046
Export shipments	20	.....	20	.....	.....	.....	.....	.....	31	.....	.....	.....
Stocks, end of month	143,049	138,928	131,833	129,701	130,155	130,666	130,865	129,825	129,886	129,506	129,451	132,025
World Production Rate (Daily Average)												
Copper	4,209	4,224	3,919	3,924	4,023	3,940	4,070	3,940	.....	.....	.....	.....
Lead	3,988	3,995	3,691	3,906	3,654	3,806	3,980	3,976	3,938	3,872	3,598	3,621
Zinc	3,212	3,127	2,860	2,766	2,814	2,736	2,725	2,601	2,600	2,615	2,623	2,528
Tin (Metric Tons)	426	439	401	416	349	352	325	351	311	260	347	.....

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